

Metallurgical & Chemical Engineering

Volume XIII, Number 15

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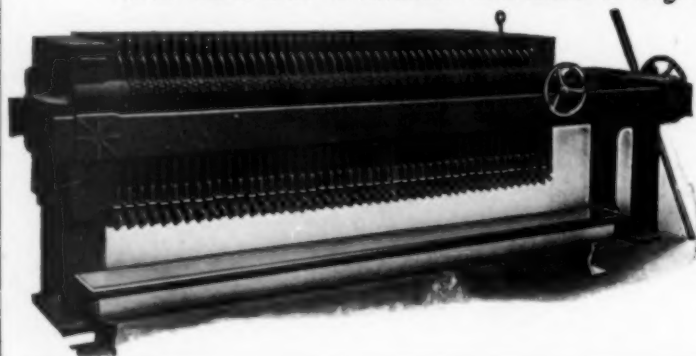
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Flow-Sheets of Organization

In his opening address as Honorary President of the recent International Engineering Congress, General Goethals enlarged on the subject of organization as a factor in carrying engineering projects to a successful issue. The ability to create and maintain a loyal and efficient organization of the human element in any project is one of the first essentials to success. With it there is harmony and economy; without it, confusion and waste.

Borrowing an analogy from a well-established metallurgical custom, a flow-sheet of organization will be found quite as useful in visualizing the proper relations between the different departments of a company, as a flow-sheet of ore-treatment is in illustrating the different steps in a process. The very act of reducing the details of an organization to diagrammatic form will clarify the scheme in the mind of the chief, crystallize a more efficient method of transacting business through subordinate channels, and establish responsibility throughout the entire force.

It is a well-recognized and fundamental principle of organization that definite authority must be delegated if responsibility is to be demanded, just as in military affairs rank is established to insure command. Once established, the integrity of the organization must be maintained, and loyalty becomes the important consideration. The individual must recognize that the organization represents a purpose or object greater than any one factor in it, which can be attained only by harmonious and co-operative effort.

The more diverse the activities of an organization the greater the need for a well-established flow-sheet of authority and responsibility. For instance, some mining and metallurgical concerns carry on the three operations of mining, concentrating and smelting. It is self-evident that each of these operations is but a single factor in the accomplishment of a certain purpose, and ultimate success requires the proper correlation of work in the several departments. Loyalty demands that, so far as possible, the mine shall meet the requirements of the mill, and the mill of the smelter. Any attempt on the part of one to make a record at the expense of the other defeats the ultimate object.

Within a department the status of the individual must be recognized by his subordinates and superiors, orders being transmitted down the line of authority, and grievances, complaints and suggestions upward through the line of responsibility; provided, that in the latter case a channel be open to the head of the organization if the established flow-sheet is rendered

ineffective for a subordinate through unfairness on the part of an immediate or intermediate superior.

All these and many other features of organization and management will become apparent, and their importance more fully realized, if delineated in the form of a flow-sheet. Every factor in the organization will understand his position and appreciate his importance, with the result that an *esprit de corps* will be created that will overcome many obstacles tending to wreck an otherwise well-conceived and meritorious project.

Productive Capacities in the Iron and Steel Industry

The present general advance in iron and steel-market prices began at the end of last December, for on the first day of this year the regularly quoted market on bars, plates and shapes advanced from 1.05 cent to 1.10 cent. Throughout the first half of the year these commodities, and practically all other finished steel products, advanced in a moderate way. The average advance was about \$2 per ton. The demand was steadily increasing, though it was not until July or August that it was sufficient to engage the full capacity of the steel industry. During the first half of the year, however, there was no general advance in pig iron.

By the first of September steel, merely as steel, had become scarce, while pig iron was still plentiful. The condition was precisely the one observers had not expected. Since 1912, when pig iron and steel prices were fairly well aligned with each other, there had been much more new construction of steel-making capacity than of blast furnaces, and from this showing pig iron should have grown scarce first. Like many other things, a condition could be explained when realized, although it could not be predicted. The blast furnaces individually were making better records than ever before, and the country's capacity, based on the performance of individual furnaces in the past few months, appears to be nearer 40,000,000 tons a year than the 35,000,000 tons estimated not so long ago. Furthermore, the demands of the foundry trade were light and more merchant furnaces served the steel instead of the foundry trade.

A new turn seems to have been given the kaleidoscope, and while steel prices have not ceased advancing by any means the pig iron market is now advancing very sharply. It is estimated that the average advance in pig iron during November has been not less than \$1.50 a ton, whereas during all the movement that preceded there was an advance of only about \$2.50. Nevertheless the foundry trade is still not busy and there is reason to believe that should the buying of foundry pig iron become as heavy as has been the buying of steel the demands upon the merchant furnaces could not be met. Blast furnace capacity might then be found to be deficient with respect to steel-making capacity, which would fit expectations entertained until recently.

There is so much news in the iron and steel trade, even in dull times, and events move so rapidly, that it is seldom long range comparisons are made. Comparison of pig iron and steel prices suggests a very interesting one. Pig iron has now advanced about \$4 a ton, while finished steel prices have advanced more than \$10 a

ton. It is not more than ten years ago that in most quarters the impression prevailed that the strength of a large steel-producing interest lay chiefly in its control of raw materials, and iron ore in particular. Should materials become scarce those that lay closest to being in the raw stage should advance the most. Such was the case in the market movement of 1905-6-7, when Bessemer pig iron advanced to \$23.25 and basic to \$23 at Valley furnaces, while the large mills did not quote above 1.70 cent for plates and shapes and 1.60 cent for bars. On Nov. 18 last the Carnegie Steel Company advanced its price on bars, plates and shapes to 1.70 cent, but on that day Bessemer pig iron was \$16.75, Valley, and basic iron \$16. The spread between pig iron and finished steel was \$7 wider than was the spread when the 1907 market was at its top. Ore is less important and steel making and finishing capacity more important.

Flotation and Cyanidation

The success of flotation as a concentration process is bringing in its wake a number of interesting problems in metallurgy, not the least of which relates to the cyanidation of flotation products—concentrates and tailings. In fact the future development of flotation may have an economic bearing on the continued use of the cyanide process in some places.

It has long been customary in many cyanide mills to combine table concentration with cyanidation for a variety of purposes: the removal of a refractory portion of the ore as a concentrate for smelting, followed by cyanidation of the tailing; the production of a barren tailing which is discarded, while the concentrate is cyanided; or, the separation of a concentrate for re-grinding, after which it is again combined with the tailing for cyanidation. These processes have become standard and their details are well understood.

With the introduction of oil flotation a question naturally arose as to the possibility of cyaniding its products just as table products had been treated. At present it does not seem at all certain that the idea is easy of general accomplishment. In some cases it has been possible to substitute oil flotation for gravity concentration and cyanide the concentrates without difficulty. On the other hand, however, one metallurgist experienced in cyanidation has said that thus far he has been unable to cyanide raw flotation concentrates. And in another case where table tailing is being successfully treated by cyanidation, it has been impossible, at least in preliminary tests, to cyanide the tailing from the same ore when concentrated by flotation.

The difficulty is not regarded as insurmountable, and in all probability the problem will be solved in the extended research that is now under way. A rare opportunity is offered to study the physical and chemical effect of a small quantity of oil to see how far it may inhibit the chemical and electrochemical phenomena involved in cyanidation. It is not in concentrates alone that oil may cause trouble, for in the use of the more water-miscible oils, such as turpentine, appreciable quantities may exist in the tailings. Again, it is possible, through selective action, that in the treatment of

some ores flotation may fail to remove as a concentrate the refractory portion which gravity concentration handles successfully, thereby leaving in the tailing a valuable constituent not amenable to cyanidation.

With regard to flotation as a menace to the continued success of cyanidation in some districts, it has been suggested that the latter process may find it difficult to maintain permanently its present supremacy. The economic balance may swing in favor of a combination of flotation and smelting, as against cyanidation. Capital and operating cost would seem to favor flotation, and with a normal recovery and favorable smelting rates it is quite possible that flotation could appear as an active competitor to cyanidation. The cost of flotation oils is increasing, due to the unusual demand for tar products and the limited production of wood oils. But the oil cost is not the major item in the total cost and an increase in price will not be prohibitive.

Imports and Exports of Steel

Few industries have experienced such changes in the character of their foreign trade as has the iron and steel industry, and quite naturally, since substantially all the countries apart from the United States that produce iron and steel are at war with each other. While iron and steel is one of the most nearly self-contained of all industries it is not wholly so, and thus the country that exports some descriptions of material usually imports others. The United States, for instance, has had its imports of iron ore cut down. The latest import and export statistics available are those for the month of September and the nine months of the calendar year ending September, hence a comparison, to illustrate war conditions, is naturally made between the first nine months of the years 1913 and 1915 respectively. There was no very great difference in the industrial activity in the two periods, the production of pig iron in the United States being 15 per cent less in the later than in the earlier pig iron. Imports of iron ore, however, decreased from 1,916,733 tons to 971,098 tons, or practically 50 per cent, there being decreases in imports alike from Sweden, Spain, Canada and Cuba. There were also great decreases in imports of pig iron and ferro-manganese. Imports of unfinished steel, chiefly high-grade alloy steel, decreased from 18,000 tons to 9300 tons. Imports of machinery, cutlery and the like suffered large decreases.

Our exports of iron and steel and manufactures thereof are always vastly greater than our imports of corresponding material, the calendar year 1913 showing imports (not including iron ore) of \$33,601,985 against exports of \$293,934,160, nearly nine times as great. Naturally one expects to find great changes in the iron and steel export trade, and a glance at the statistics does not disappoint him. By setting in antithesis some of the items one may see quite clearly a few of the changes that war has wrought. Comparing, as before, figures for the first nine months of 1913 and 1915 respectively, exports of agricultural implements, hitherto taken largely by Europe, have decreased from \$27,622,594 to \$10,371,379, while exports of breadstuffs have increased from \$166,-

848,067 to \$423,360,972. Europe must have the food ready to eat. The wheelbarrow yields to the motorcycle, the wagon to the motor truck and the carriage to the passenger automobile. Exports of wheelbarrows, push carts, etc., decreased from \$598,648 to \$232,391, while the number of motorcycles exported increased from 3459 to 11,859; wagons exported decreased from 16,980 to 5949, while motor trucks increased from 778 to 17,269; carriages decreased from 13,458 to 1139, while passenger automobiles increased from 20,175 to 31,036. This is a modern war and there is nothing slow about it. Exports of railway cars to North and South America and Japan decreased from \$12,883,673 to \$1,234,168, or 88 per cent, while exports to "other countries" undoubtedly Europe almost exclusively, increased from \$618,773 to \$1,721,620, or 180 per cent.

Exports of cash registers decreased from 33,064 to 11,920, but horseshoe exports increased from 950 tons to 14,000 tons. Printing press exports decreased from \$2,031,774 to \$1,065,529, but exports of metal working machinery increased from \$11,980,778 to \$30,844,199. From the ordinary steel mill products antitheses can readily be picked out. Steel bars, used chiefly for the manufacture of shells, increased from 164,000 tons to 300,000 tons, while structural iron and steel, not particularly useful for war purposes, decreased from 309,000 tons to 169,000 tons.

The government export statistics customarily give the principal countries of destination in the case of important exports, but as the classification was established before the present conditions arose it is not well fitted for the presentation of illuminating statistics. In the case of plain and barb wire, for instance, the adopted classification is: Canada, Mexico, Cuba, Argentina, Brazil, other South America, British Oceania, British Africa and "other countries." The exports to the countries named increased from 124,000 tons to 145,000 tons, their total imports being undoubtedly decreased, but a much larger proportion being drawn from the United States, while the "other countries" increased their imports from 18,000 tons to 196,000 tons. There is reason to infer that plain and barb wire has been taken for war purposes at the rate of 225,000 or 250,000 tons a year.

Exports of such iron and steel commodities as are returned by weight, including scrap, pig iron, unfinished steel, rolled iron and steel, cast and wrought pipe, nails, wire, etc., but no machinery, have been as follows in gross tons:

1912	2,948,466
1913	2,730,681
1914	2,085,511
First quarter, 1915	458,470
Second quarter	842,291
Third quarter	1,166,667

Thus these exports have reached a rate of 4,650,000 gross tons a year. Iron and steel used in the manufacture of shells, machinery, automobiles, railway rolling stock, etc., undoubtedly amounts to more than the equivalent of 2,400,000 tons a year, so that iron and steel to the extent of more than 7,000,000 tons a year is involved in the direct and indirect exports.

Readers' Views and Comments

Wet Ores in Charcoal Blast Furnaces

To the Editor of Metallurgical & Chemical Engineering:

SIR:—I read with a great deal of interest Mr. Johnson's working out mathematically the effect of wet ores in charcoal practice. The actual fact that filling wet ores was injurious to the efficiency of charcoal furnaces both in output and fuel consumption has been a well-known fact ever since iron was made with charcoal in blast furnaces, and the main reason, viz., that the stack heat was lessened, was also perfectly well understood.

In my own experience, when quite a young man, I often had occasion to notice the tremendous difference in furnace working between wet and dry ores.

One of the furnaces under my charge was a small cold-blast charcoal furnace using soft fossil ores from the company's mines. An attempt was always made to have a sufficient supply of ore on hand for six months' use at least. This ore was unloaded from the mine cars on piles, not over 6 ft. or 8 ft. high, and under a shed with open sides. The ore merely being protected by a roof.

The ore in the course of time became air-dried, and the moisture would not average over from 4 to 6 per cent. The dried ore itself averaged about 45 per cent in iron with 15 to 18 silica and 4 to 6 alumina.

The burden was 600 lb. ore, 30 to 40 lb. limestone on a charcoal unit of 18 bu., or 360 lb. well-mixed coal.

Sometimes, however, the ore stocks would run out and we would be compelled to use ore direct from the mines, often running as high as 15 and even 20 per cent of water. In such cases the burden would have to be cut to 525 and even as low as 500 lb. with a corresponding reduction in iron, and the iron generally a grade or two lower than made with the full burden of dry ore. I have often seen the temperature of the escaping gases drop to 120 deg. Fahr., so cool, in fact, that the hand could be held in the gases issuing from the open top. For they were so wet that they would not ignite until they went into the boiler combustion chamber, where a couple of logs of wood had generally to be kept burning to light the wet gases.

This was the common practice and experience all over central Pennsylvania where fossil ores were used, and I remember seeing at Mount Alto several large ore roasters which the superintendent, Col. John Weistling, had built to dehydrate the ordinary brown hematite (limonite) ores used at this plant. These ores were the average brown hematite found throughout Pennsylvania and Virginia running after washing about 40 to 45 per cent in iron, and containing 10 to 13 per cent combined water, very little if any sulphur, and no carbonic acid. He roasted these ores simply to drive off the moisture, as they were amply porous for easy reduction, and he told me he had found by experience that the use of roasted ore made a reduction of from 10 to 12 per cent in his fuel. It was also a common practice throughout central Pennsylvania at the old charcoal furnaces to pile the brown hematite ores in piles sometimes several hundred feet long, 8 to 10 ft. high and 20 ft. wide. These piles were formed by a layer of cordwood and kindling on the ground, then about a foot of ore, 2 to 3 in. of charcoal breeze, 1 ft. of ore and so on to the top. The pile was lit at one end and used while piling was going on at the other. Whenever roasted ore ran out and raw was used it was necessary to cut the burden.

Pennsylvania Steel Company,
Lebanon, Pa.

R. H. LEE.

The Electric Furnace in the Foundry

To the Editor of Metallurgical & Chemical Engineering:

SIR:—For some time past, the various technical journals have contained articles on "the electric furnace in the foundry." Each author of these articles has given his opinions and experiences, with the uncorroborated experiences of others.

But with all this, and in summing up the substance of the literature the vital question stands out, "Why is electric-process steel superior to steel made by other processes?"

To answer this question we can find a considerable amount of literature. Good authorities say that the conditions of melting and refining in the electric furnace, out of contact with the flame and gases, and in a neutral or reducing atmosphere, contribute to the superiority of the product. Others state that the high temperature and the reducing conditions make a superior product. Numerous convincing reasons are given by the manufacturers of electric-steel furnaces. Among them we can cite the following from their advertisements in the technical journals: "Crucible steel quality, with maximum of uniformity," "will produce high-grade tool steel from pig and ore," "excellent castings," "low cost of upkeep in producing every grade of iron and steel," and many more that have the same tone. On the other side the operators have arranged themselves, with an equally imposing lineup of statements. As their argument for superiority of electric-process steel they give: "Homogeneity," "absence of segregation," "elimination of oxides," "complete control of elements," "great tenacity to finished product," "high elastic limit," "high ratio of elastic limit to ultimate strength," "ready response to heat treatment," "ability to obtain very hot metal," and many more that I have not compiled.

In the capitulation of all this literature, it appears that the manufacturers of the electric furnaces, and the operators of the electric process have come to the same general conclusion, namely, that "the electric process for steel outclasses anything yet produced for an as-near-to-perfect product."

My remarks might have the tone of an antagonist to the electric-steel process. But quite to the contrary. What I am endeavoring to bring out is the point, why the electric-steel process *necessarily* should outclass anything ever produced in an as-near-to-perfect product.

Primarily, the melting chamber is enclosed from outside atmospheric effects. The melting down is carried on in a slightly oxidizing condition. The refining is decidedly reducing and neutral. No oxidizing gases have contact with the bath, as in the processes using a gas flame for melting. The protection of the metal from contaminations taken up from the gases, as in gas-fired furnaces. The electrodes give no contaminating elements to the metal. The temperature can be controlled by regulating the amperage going to the furnace. These facts from a physical standpoint of advantages of the apparatus have a great effect on the chemical properties of the product.

Some operators and owners of the electric furnace for steel have made statements that "they did not have the anticipated results from the furnace they put in their plant." Others said "they had trouble in understanding the electrical machinery connected with it." And still another stated "he was always afraid some-

thing was going wrong with the juice." One owner told me it was difficult to sell their castings, when another process at less cost was in the field, making cheaper castings. As a general thing, users of steel castings and steel are not looking for something cheaper, but something better, to fill their requirements.

The first essential in any furnace is the proper design of the apparatus, to carry out economically and satisfactorily the duty imposed upon it. Most foundrymen are men of limited technical training, and do not readily take up a complicated contrivance, especially an electrical one on a furnace. This feature is a very commendable one to the manufacturers of electric furnaces in America. Simplicity of the American-made furnace is characteristic. With such an apparatus the foundryman can manipulate the furnace without days or weeks of instructions.

A furnace might be supplied to a customer to handle a certain maximum size scrap. A large supply, or a large field to draw from, might exist at the time the furnace was installed. Eventually this supply of scrap is consumed, and a less desirable material must be utilized for the furnace. Possibly furnace alterations become necessary, causing an extra expenditure to the owner. The operator may not be able to operate his furnace as he did when he had the desirable and satisfactory scrap available. His costs increase, other irregularities present themselves such as, time to charge up, melting down, refining. Indirectly he finds that his production has decreased slightly.

An instance of this took place at a plant at which I was. The furnace was originally built to handle scrap of a certain maximum size. This pertains to the size of the charging-door openings. Small shovel scrap and such scrap as could be charged easily, had been available for some time. Eventually this desirable scrap became depleted. The management purchased an 80-ton lot of scrap split-steel gears, weighing approximately 120 lb. each, and 20 in. in diameter. Each gear was bolted with two 1½-in. steel bolts. The gears as they were were too large for our furnace. Consequently each gear had to be cut apart by taking off the two nuts. This necessitated extra expense in handling the scrap; the large chunks took a longer time to charge; the time of melting down was increased, and the output of the furnace for a given time was decreased. The cost per ton increased. Here the proper scrap was the vital point in the economy of the operations.

Right here I wish to state that, in my opinion, the "sesame" to making high-grade superior steel by the electric process, both economically and satisfactorily, is dependent on the scrap and the purity and condition of the materials charged into the furnace.

In using the all-scrap process, which I consider the simplest of processes for electric-steel castings, the scrap must be given careful attention. In packing crucibles for the crucible process, only the cleanest and most desirable selected material was used. Punchings of medium open-hearth steel were riddled to take out all dirt and loose rust. Manufacturers of crucible steel made a special iron for their crucible plants, to get the best materials for the process. Careful supervision during melting and careful selection of the materials charged produced a high-grade superior-quality steel. Up to the advent of the electric-furnace process, crucible steel was considered without a competitor. But without the careful selection of the melting scrap, crucible steel would not have been what crucible steel is known to be.

Electric steel now has the same chance that crucible steel had. The materials must have the necessary attention to guarantee a product from them. Unless the cleanest and best selected scrap is utilized for the elec-

tric furnace, electric-process steel will not fulfill the claims made for it.

When a manufacturer of castings by the electric-furnace process made the remark that he had not found the electric furnace what had been claimed for it, I questioned him on several points. Among them I asked to see his scrap. He showed me his stocks. The purchased scrap was out in the weather, coated with rust and dirt. Layers of rust peeled from the material. His scrap castings were sandy and dirty. The fillets of the castings contained considerable sand, as well as sand sticking all over them. The sprue was likewise sandy and unclean. Core sand was still in the cored holes of the castings. And this scrap went to the furnace just as I saw it. The punchings, of which there was a large stock, were shoveled from the floor, and, sand and dirt and rust mixed with them, taken to the furnace.

With such materials charged no process will turn out a high-grade superior product, especially what is desired for a steel casting. Every bit of dirt put into the furnace must rise through the molten bath and be taken off by the slag. Unnecessary dirt charged takes unnecessary time to finish the heat. Time in the electric process means power consumption, and power consumption means dollars and cents. Eventually a heat is poured exhibiting dirt holes and entrained slag in the castings, and the furnace and the process is condemned. The presence of iron oxide, as rust on the scrap charged, increases the labor of "killing" or deoxidizing the metal. Ferrous oxide is very soluble in a melted steel bath. During the melting down of a cold charge of scrap, oxidation of the metalloids and metal takes place. These metallic oxides must be reduced, the solid product brought to the surface and the gaseous products allowed to escape. By adding an unnecessary amount of oxides from the scrap an unnecessary amount has to be reduced in cleaning the metal. This increases the time of making a heat, because unnecessary labor and time has to be expended to get the metal ready to pour. As the atmosphere of the melting chamber during melting down of the charge is oxidizing, sufficient oxidation goes on to oxidize phosphorous, etc., without adding any outside oxidizer.

By proper protection against the weather, sand blasting of the extremely dirty and heavy pieces, and rattling of the lighter shovel scrap, a clean and acceptable charge can be provided for the furnace. This might militate against costs. But with a superior product a better price can be obtained for it.

Absence of segregation, elimination of oxides can be controlled to a considerable extent by careful observance of the scrap charged. The presence of manganese sulphide has considerable to do with the defects of segregation. Much of the so-called "blowholes" are due to the presence of manganese sulphide. The oxidation effect and the presence of oxides might exist after the metal has been tested and found to be deoxidized. The extremely high temperature of the electric furnace causes the metal to become considerably superheated. At the time of pouring from the furnace into the ladles a state of oxidization could exist, in which the metal becomes oxidized and gets "wild." Then the purging of the molds shows a gas evolution in the metal.

In conclusion, I consider the use of clean scrap to be the most essential and primary factor toward turning out a superior and high-grade product from the electric steel furnace. With the small capacity per heat, and the necessity of selecting clean and desirable scrap for each heat, the electric-furnace process for steel making can be made to give a product superior to anything on the market, and at a price that can compete with other processes.

R. C. GOSROW.

Heroult, California.

Award of Nobel Prize in Chemistry to Theodore W. Richards

The award of the Nobel prize for 1915 to Prof. Theodore W. Richards of Harvard University has been announced. The work of Professor Richards on the atomic weights is well known and is generally recognized as the best work which has been done on this subject. With the help of assistants he has so far revised the atomic weights of about twenty elements. The results were published in various papers, published in the *Journal* of the American Chemical Society and elsewhere, and in a book "Experimentelle Untersuchungen über Atomgewichte" 1909. He has made many other contributions to physical chemistry, thermochemistry and electrochemistry. Professor Richards has been connected with Harvard for a long time, having taught there since 1889. In 1901 he was offered a chair in chemistry at the University of Göttingen, Germany, but declined, deciding to stay at Harvard. In 1907 he was visiting professor at the University of Berlin and in 1914 was president of the American Chemical Society.

The Failure of the Hull of the Sea Call

In view of the publicity which has been given the failure of the hull of the splendid schooner Sea Call, belonging to Alexander Cochran of Yonkers, N. Y., it is timely to have a statement from the Supplee-Biddle Hardware Company, Philadelphia, Pa., dealers in Monel metal, as the hull of the Sea Call was composed of Monel metal below the water line. This statement of the causes of the failure, given in the November number of the Supplee-Biddle *Bulletin*, points out that the failure was due to electrolytic action between steel as negative pole and Monel metal as positive pole.

Steel plates were used above the water line, and by mistake a few steel rivets were used in some of the Monel metal plates below the water line. It was the eating away of these rivets which first showed the corrosion that was taking place. The Monel metal was not affected. The most rigid examination was made of this hull, in view of the importance of the question, and it was concluded that had the hull been made only of Monel metal corrosion would not have taken place.

The hull was hauled out of water on Aug. 24 in the presence of a number of engineers representing prominent builders and owners.

The most exhaustive examination was made by these gentlemen to satisfy themselves and the companies they represent of the facts. It was therefore gratifying to every one interested in the production and use of Monel metal to know of the perfect condition of Monel metal in resisting the galvanic action that had destroyed steel

and rendered an investment said to be half of a million dollars practically worthless.

The examination showed that the steel section under the water had seriously corroded. One of the most noticeable instances was that of the fine steel rib of the rudder frame. The interior framing of the vessel was of steel and this was, of course, connected with the Monel metal plates which made it possible that under the action of the bilge water corrosion would extend to all this steel work until the inner frame of the yacht was in a dangerous condition.

If the fault lay with Monel metal and there was danger of corrosion and rusting away it can readily be seen that its future as a roofing and as a general utility metal would be killed. It was on this account that the exhaustive study of the case was made to determine exactly where the responsibility lay. It is therefore gratifying to its makers and to the judgment of the various engineers concerned, that the cause of the trouble and resulting loss was found entirely outside of this valuable natural alloy and due alone to the combination of steel with the action of salt water.

Coming Society Meetings

A joint meeting of the New York Sections of the American Chemical Society, American Electrochemical Society and Society of Chemical Industry will be held in Rumford Hall, Chemists Club, on Friday evening, Dec. 10. The subject will be a continuation of that taken up at the last meeting of the American Chemical Society and also of the Society of Chemical Industry, viz.: Research and Chemical Industry. This will be Boston "Tech" night, the first A. C. S. meeting having been given over to Columbia University.

The winter meeting of the American Institute of Chemical Engineers will be held in Baltimore, Md., Jan. 12-15. A number of excursions to some of Baltimore's important industries will be arranged and the experimental laboratories of Johns Hopkins University and the U. S. Naval Academy at Annapolis will be visited.

The second Pan-American Scientific Congress will be held in Washington, D. C., Dec. 27 to Jan. 8. One of the nine sections will be devoted to mining and metallurgy, economic geology and applied chemistry.

The spring meeting of the American Electrochemical Society will be held in Washington, D. C., from April 27 to 29, 1916.

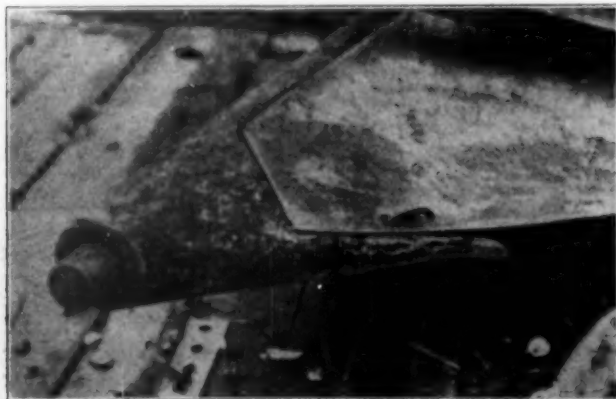
Electrochemistry and Lighting

Joint Meeting of New York Section of American Electrochemical Society and Illuminating Engineering Society

A very interesting joint meeting of the New York Section of the American Electrochemical Society and of the Illuminating Engineering Society was held on Thursday, Nov. 11, 1915, at the United Engineering Societies Building in New York. The program of the evening was electric conduction through gases in its relation to light production. Dr. Colin G. Fink, of the Edison Lamp Works, Harrison, N. J., chairman of the New York Section of the American Electrochemical Society, presided.

UNSTABLE STATES OF THE ARC

The first paper was presented by Professor W. G. Cady, of Wesleyan University, on "Unstable States in Arc and Glow." This paper, which we published in full in our issue of Nov. 15 (page 866), was illustrated by lantern slides and at its conclusion Professor Cady



SECTION SHOWING STEEL PITTED AND MONEL SHEET UNINJURED

made some very interesting experimental demonstrations of the singing arc, showing the possibilities of exact tuning. The paper was discussed by Dr. Moore, Mr. Mott and Dr. McKay.

A TRUE DAYLIGHT ARC LAMP

Mr. D. McFarlan Moore of the Edison Lamp Works, Harrison, N. J., followed with an elaborate and long paper on the latest developments of Moore tube lamps. One of his most interesting exhibits was a *neon tube lamp*, but the main part of the paper dealt with his *carbon dioxide tube lamp for color matching*. He employs in it a new principle for automatically replenishing the supply of gas within the tube as the gas is used up during the operation of the lamp.

The application of this lamp was also illustrated by showing the difference in the color value of objects such as dress goods, silks, meat and flowers when illuminated first solely by the light of the standard tungsten lamp and then solely by the light of the Moore daylight tube lamp. In the course of his paper Mr. Moore called attention to the possibilities of the development of gaseous conductor lighting and the limitations of solid conductor lighting.

Elaborate spectrophotometric investigations have been made to show that articles of various colors, when viewed solely by the light of the new tube lamp, have exactly the same shades of color which they display when viewed by the light from a clear sky. Dyers and other color experts thoroughly agree that the standard light which they wish for color judging is that which enters a window from a clear north sky at an angle of about 45 deg., and preferably about mid-afternoon with a clear sun shining in the south.

There is reproduced in Fig. 1 the spectrophotometric curve of the CO₂ spectrum, which consists of a broken line with many jagged peaks, but an average of these peaks results in a smooth curve which gives a correct idea of the effective color characteristics so that it should be used for comparison with other line and continuous spectra. The effect of the banded character of the CO₂ spectrum, the tabulation of which consists of a broken line, was thoroughly investigated to determine its relation to the color-matching properties of the lamp.

The following samples of colored silk taken from the National Color Card of America were selected to represent the various parts of the spectrum and were spectrophotometrically tested to determine what the actual color composition was: violet, national blue, Yale blue, emerald, lemon, orange and scarlet.

The result of each of these tests was carefully plotted in the form of a curve, and a review of the data obtained in connection with these reflection curves showed that the irregularities in the CO₂ spectrum were absolutely without visible effect, and that they deviated only very slightly from north-sky values, to which they were compared. But it may be pointed out that if these north-sky values had all been taken at an angle of

about 45 deg. there would have been no detectable variation, and that therefore the color-matching values of the CO₂ spectrum were all that could be desired.

The straight-tube lamp is contained in an elongated sheet-metal case, which, however, is provided with a screw base similar to that used on the larger sizes of incandescent lamps. Instead of the lamp being fed CO₂ gas by means of an auto-magnetic feed valve, the gas is generated automatically within the tube itself. Near each electrode is placed a small bulb about an inch long, containing calcium carbonate, from which emanates the carbon-dioxide CO₂ when the resistance wires embedded in it become heated to the proper degree by reason of their being connected in shunt to the gas column. The gas column measures $\frac{7}{8}$ in. in diameter and 1 ft. in length, and appears as a solid bar of light of intense whiteness.

The foot-candles available near the tube total more than 200, making this apparatus suitable for the very closest color discriminations. The degree of vacuum is automatically held within 0.001 mm. of mercury column, and the apparatus will stand wide line-voltage fluctuations without becoming disarranged in any way.

This new lamp is suitable for making absolutely correct color determinations, and while it is applicable to a very wide field, it is particularly useful in enabling the dye shops of the great textile industry to run night shifts.

Since light is the source of all color, it has been proposed that this new lamp receive official recognition as a color unit from the International Conference on Electrical Units and Standards.

A NEW ARC LAMP WITH UNCONSUMED ELECTRODES

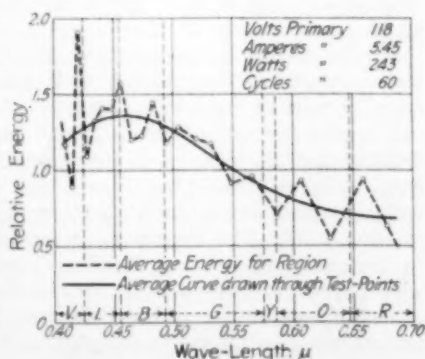
The last paper of the session was presented by Mr. W. A. Darrah, of the Westinghouse Company, on "The Electric Arc in Vapors and Gases at Reduced Pressures." The idea of the lamp seems fundamentally new as it is a flame arc lamp between tungsten electrodes which are not consumed. The light-giving qualities are obtained from special vapors introduced into the lamp. The paper was illustrated by lantern slides and experimental demonstrations, and was discussed by Dr. Moore and Mr. Mott. Mr. Darrah's paper is printed almost in full elsewhere in this issue.

Co-operation Between University and Industry

Meeting of New York Section of American Chemical Society

The first of a series of meetings to be held by the New York Section of the American Chemical Society for the discussion of the problem of co-operation between university and industry was held in Rumford Hall, Chemists' Club, Friday evening, Nov. 12. The attendance was very good and the hall was well filled.

After a short business meeting Dr. T. B. Wagner, chairman of the New York Section, said in his introductory remarks that research has not been developed in this country to the same extent as teaching. Even though many of our large establishments maintain finely equipped research laboratories we must look to the university for the training of the young man in research. Further activity in this work on the part of the universities is much needed. The far-reaching potential influence of university on chemical industry was forcibly brought to his attention when some years ago he visited his old teacher Emil Fischer and upon being shown into his study, he found the professor discussing an important industrial problem with the director of one of Germany's largest chemical concerns.



CURVE SHOWING DISTRIBUTION OF ENERGY IN SPECTRUM OF CARBON-DIOXIDE TUBE LAMP

The speakers of the evening, President Butler and Professors Whitaker, Gies and Lucke, all of Columbia University, told of the work done there to aid the industries and of the further work which could be done. Their remarks were not intended, however, to apply only to Columbia but to every university.

President **Nicolas Murray Butler** of Columbia University, recalled with pleasure the chemical lectures of Professor Chandler he attended thirty-five years ago. He hesitated, however, to dwell upon the chemistry he learned in those early days fearing that the audience might regard it as archeology rather than chemistry. Chemistry has undergone many great changes in the last thirty-five years and the present European war has brought to the attention of the world the various stages of development in the different countries. In recent years we have heard much of the development and conservation of our natural resources, but not until this country felt the shock of the European war did it begin to realize the importance of organizing its forces, scientific and natural, for the strength of the nation.

Dr. Butler said great harm had been done in this country through the misuse of the word university. Any three men could get together in most States, except New York and Pennsylvania, he said, and form a university, so that the name has not had the proper significance to the general public and its aims and functions were little understood, consequently co-operation between the university and industry was not to be expected. But we are now beginning to see that a university is not only a place where persons are taught, but it is a public service institution. In the secondary schools and colleges knowledge is imparted to the young men, but in the university school emphasis is laid upon the extension and application of knowledge and upon the service of the public. A real university must be inspired and directed by the spirit of investigation and research.

Dr. Butler emphasized the performing of practical research. "Why do things for exercise in a university when there are so many practical questions waiting to be taken up." In this connection Dr. Butler told a story of a farmer digging a hole in front of his house and filling it up again for exercise while hiring a man to dig a trench for him. The university wants to keep down to a minimum the contrast between the academic and the practical. The university needs contact with and the co-operation of organized bodies of practical men, and these men should familiarize themselves with the university laboratories.

In America there is already more co-operation between the university and the industry than in England. At the School of Architecture at Columbia there is a visitors' committee of nine. Three of the large architectural societies of this country have each appointed three members to serve on this committee. It is the duty of this committee to make periodical visits to the university, to investigate the methods of teaching and to offer such improvements as they consider advisable. A similar committee would be of great service to the School of Chemistry and thereby further the co-operation of the school with the factory. The universities are ready and willing to welcome committees of this kind.

Prof. **M. C. Whitaker** took for his subject the work of Columbia in helping the chemical industries. Chemistry is very closely related to industrial development and the function of training men is one of the most important. The training which a man receives in order to eventually become a works manager, or to hold other responsible managerial positions may be divided into two parts, the portion done by the school and the por-

tion done by the industry. The school gives the fundamentals and furnishes the student with a working basis. The industry is then responsible for allowing the man to gain the essential experience.

Professor Whitaker said the school is not able to make the rigid examination and inspection of its raw material as a manufacturer does in producing a manufactured product. The people of the industry should bear in mind that the university has no control over the "raw material" or student quality and should overlook a pardonable pride of the university whenever from rather crude material it turns out a "work of art." For this reason the yield cannot be 100 per cent from the combined training of school and industry. The industries often fail to appreciate this and expect too much. On the other hand they often fail to give the man the necessary opportunities to develop.

Industrial research has often been badly handicapped on account of unsympathetic surroundings of many of the factory laboratories. The young chemist is continually reminded of the importance of "production" and "profit" so that sooner or later he leaves his research and joins the factory force and his only chemical activity consists in the analysis of the shipments of coal.

The most important function of the educational institution is the opportunity for research. While it is true that some of our large companies have finely equipped and excellently conducted research laboratories, free from the hampering influence of the commercial production department, yet these involve a great expense and are not available to the average company. The university is the best place for research for most concerns. Professor Whitaker gave a brief outline of the course of chemical engineering at Columbia. All university research is industrial research. As regards pure research, the chief charm of it lies in its vagueness. The European war has forced us to realize that it is high time "to cash in" our scientific knowledge and make it immediately available.

Prof. **W. J. Gies**, in speaking for the department of biological chemistry said he represented the side on which both the industry and the university might help the investigator. He reviewed the work of Pasteur on the tartrates, fermentation and the French silk industry, and showed how Pasteur's researches had been of great practical application and had been suggested and helped along by his contact with the industries. He said the biological department of Columbia had been hampered in helping the industries because it had to uphold the honesty of its work. He said it was the exception rather than the rule to be asked to do honest work for some industrials. The biological department has been approached with all sorts of crooked food and medicine schemes but has had to refuse all except where a direct public benefit was to be gained. Professor Gies emphasized that it was "up to the industry. Gentlemen of the industry, it is your move."

Prof. **Charles E. Lucke**, in speaking for the mechanical engineering department said that mechanical engineering must necessarily follow chemistry and furnish the means for transferring its principles into practice. Taking the chemical industry as a whole, the equipment is largely made up of standard machinery for the different operations, such as filter presses, centrifuges, crushers, etc. The improvements arising from research in the chemical industry are not, however, all chemical. The indirect aids may sometimes be forgotten. So many contributions of the universities are indirect that they are unfamiliar to the industries and hence do not receive the proper application. New apparatus and equipment is the most important indirect contribution. The next important is fault discovery. This is a duty which

can be performed by the university investigation.

In taking up new ideas it is all important to have an experimental construction to obtain accurate data. Here is where a great barrier has arisen. Many splendid ideas have never had the necessary experimental trials for accurately deciding upon their value, and on the other hand many poor ideas have been tried and when they failed the commercial men have become discouraged and looked with disfavor upon new ideas and tests. After an experimental trial has been brought to a successful termination the final stage is the design of the apparatus. Here the university laboratory can play no part. Any correct analysis of the relations between university and industry will show that the industry is receiving both direct and indirect contributions limited only by financial support.

The chairman, Dr. Wagner, announced that the discussion would be postponed to a later meeting inasmuch as several meetings are to take up the same subject.

American Institute of Mining Engineers

New York Section

The New York Section of the American Institute of Mining Engineers, of which Mr. David H. Browne is the chairman, is arranging an interesting program for this season. The meetings will be informal dinner-meetings, affording a means of friendly intercourse, and the general tendency was well indicated by the first meeting held on Oct. 28, devoted to a discussion of the New York Subways, when the presentation of this subject by the engineers of the Subway Commission was appreciated by a large audience. The endeavor is to handle matters of general and timely interest rather than technical subjects which interest only a few. Among the subjects which will be taken up in later meetings are "The Engineer in Modern Warfare" and "The Problems of the Naval Advisory Board."

Coal and Iron in Nova Scotia and Newfoundland

Meeting of the Mining and Metallurgical Society

On Thursday, Nov. 18, a meeting of the Mining and Metallurgical Society was held at the Machinery Club, New York City. After an informal dinner, the address of the evening was made by Major C. L. CANTLEY of New Glasgow, Nova Scotia, assistant to Mr. Thomas Cantley, the president and general manager of the Nova Scotia Steel & Coal Company, concerning the coal mines of Nova Scotia and the ore mines of Newfoundland.

Major Cantley has had an acquaintance with these properties from boyhood up, not only from an office point of view, but has spent several years at and in the mines themselves, participating in the very extensive and thorough geological studies which resulted in proving definitely the extent of the ore mines, whose outcrops occur on the land, but the vast proportion of which extends out under the ocean.

Major Cantley first described, briefly, the coal mines of the Pictou region, which are not particularly large and do not differ in important parts from other coal mines, and then passed to a description of the vast coal beds at the northeast end of Cape Breton Island. These outcrop on the land, but dip at a low and uniform angle out under the ocean, the farthest commercial outcrop being only about a mile and a half back from the shore. The Dominion Coal Company and the Nova Scotia Steel & Coal Company now control practically all the important part of this field. They have each patented submarine areas many square miles in extent out under the ocean off Sydney Harbor. The commercial develop-

ment of these mines is much older than is generally realized in the United States, there having been coal mines of commercial importance in the district for over a century. By far the greater proportion of the coal mined in this field is used for bunker and commercial purposes, but the slack which results, is washed to reduce its ash and sulphur and then coked. When properly handled, the coal from some of the seams makes a good metallurgical coke.

The outcrops of the ore body are quite close to the shore of Bell Isle, a small island in Conception Bay on the extreme southeast corner of Newfoundland. The ore beds of commercial size are three in number and exceedingly uniform; even more uniform than those of the Birmingham district, as well as much richer, the ore running regularly from 50 per cent to 53 per cent in iron. These ore beds are separated and overlaid by strata of sandstone and shale, the sandstone forming the roof and greatly facilitating mining, since absolutely no timber is required. The ore beds dip out to sea at a very uniform rate of about 10 deg. and the mines are now being worked a mile and a half or two miles out under the sea.

Very careful geological examinations indicate that the quantity of ore commercially available is not less than half a billion tons and may be as high as three billion tons, constituting one of the largest of the world's ore reserves. This ore is transported to Sydney and used in the blast furnaces, of which there are nine of moderate size and good modern construction on Sydney Harbor, owned by the two companies above mentioned. The output of these furnaces is converted to steel in basic open-hearth furnaces at the two plants.

The Dominion company finishes its steel at the Sydney plant, but the Nova Scotia Steel & Coal Company ships ingots from its plant to New Glasgow where the parent works are. These include blooming mill, finishing mills and very extensive hydraulic forging plant, where forgings of very large size are produced, as well as railroad axles and the like. There are also shops for converting steel into track material, concrete bars and the like.

Just recently, a magnificent car plant has been completed and is now turning out a large number of steel and steel-and-wood cars for foreign governments.

Colonel Cantley's talk was informal and was supplemented by the replies to a great number of questions which were asked by the members.

After this technical discussion was concluded, Colonel Cantley acceded to the request of the members to tell them something of his experience in the trenches, he having been in Flanders until the late summer, when he was detailed home to assist in the manufacture of war munitions. His talk on this subject was of the keenest interest to all and a hearty vote of thanks was extended to the speaker for the pleasure and entertainment he had afforded the meeting.

The Western Metallurgical Field

Canadian Bounty on Zinc

Announcement was recently made in press dispatches emanating from Ottawa that the Canadian Government had offered a limited bounty on zinc produced in the Dominion from Canadian ores. The object of the bounty is to encourage the immediate refining of zinc within the country, and to protect producers from possible decline in the price of zinc between the end of the European war and July 31, 1917. The total amount of bounty to be paid is not to exceed \$400,000, and individual payments will be made on a sliding scale not exceeding 2 cents per pound of zinc containing not more than 2 per cent impurities, when the standard

London price for zinc falls below £33 per ton. The bounty is not payable on zinc produced before the expiration of the war or after July 31, 1915, or on zinc that is sold to the Government at a price of 8 cents or more per pound.

Zinc in British Columbia

The value of the zinc deposits of British Columbia and the metallurgical problem presented in their profitable treatment have been fully recognized for many years, at least since 1906, when Messrs. Ingalls, Argall and Gardé made an elaborate report on their investigations for the Canadian Government. Zinc in British Columbia occurs associated with lead, and has been an impediment to the reduction of the ores. Concentration has accounted for a considerable production, however, which has been shipped to the United States for smelting. Electric smelting has been tried, a plant having been erected at Frank, Alberta, in 1904, and at Nelson, B. C., in 1908. Further experiments were made at McGill University in 1910 and again at Nelson in 1913, but none was commercially successful, mainly due to difficulties in condensation of the zinc vapor.

At the present time the problem is being attacked along different lines, looking toward a hydrometallurgical treatment of the ores and electrolytic reduction of zinc. Different processes are being tested, but in general they involve the roasting of the zinc-lead-silver ores, extraction of the zinc by leaching, electrolysis of the lixivium and smelting of the lead-silver residue. At Trail, B. C., the Consolidated Mining & Smelting Co. is using sulphuric acid as the solvent and electrolyzing zinc sulphate solution. The Weedon Mining Co. is using the Watts process in a new plant at Welland, Ont., and the Standard Silver Lead Mining Co. has adopted the French process at its works at Silverton, B. C.

The French Process for Electrolytic Zinc

The process of A. Gordon French was recently described in the *Bulletin* of the Canadian Mining Institute, in an excerpt from an article contributed by a metallurgical engineer to the *Nelson Daily News*. "The method consists in the use of a solution of bisulphate of soda, which is a cheap waste product, and a small quantity of manganese—an ingredient which occurs with most zinc ores. Bisulphate of soda dissolves the zinc from the roasted ore almost instantaneously. Along with the manganese in solution it completely prevents the anodes from being affected and after years of constant use they are as good to-day as at the beginning. This solution also has very little resistance to the passage of the electric current in the electrolytic vats and the consumption of current is thus lowered. It has also the peculiarity of throwing out of solution practically all the impurities which usually contaminate the zinc. Early in 1912 the Consolidated Mining & Smelting Company of Canada acquired the sole rights for the use of this process in Canada. It was only toward the end of last year that they had partially erected a plant of somewhat large dimensions, and after they had produced a few tons of zinc the contract between French's company and the Consolidated was canceled.

"A demonstrating plant was then erected at Silverton, Slocan Lake, by the Standard Silver-Lead Mining Co., on a large enough scale to thoroughly prove that the process would work equally well on a very large scale. It comprised two completely equipped dissolving vats, each capable of dealing with 2500 lb. of liquor at a time and about 1000 lb. of ore. The zinc-depleted liquor containing bisulphate of soda from the electrolytic vat is pumped into the upper of these two vats. There is then added 700 to 1000 lb. of roasted ore from which the zinc

has been partially extracted in a previous operation. In about an hour solution of the zinc is finished and when the liquor has settled for a short time it is allowed to run down into the second dissolver, placed at a lower level. The sludge remaining in the bottom of the dissolver is then pumped through a filter press to remove the water and the residue contains all the silver and lead. As the sulphur to the extent of more than 20 per cent has been driven off in the roasting operation and zinc to the amount of about 40 per cent has been extracted, the residue of the ore, of which a large quantity was treated, only weighed about 45 per cent of the original ore used. As it contained all the silver and lead originally in the roasted ore, the percentage of these metals was more than doubled. For instance, the ore treated had 31 per cent zinc, 3.5 per cent lead, and 32 oz. silver per ton. The zinc depleted residue contained 76 oz. silver per ton, and 8.7 per cent lead, and 92 per cent of the zinc had been taken out, i. e., 2.7 per cent of the zinc counted on the original ore used was left in the residue. This residue is now ready for smelting or other treatment for the recovery of its lead and silver in the usual way.

"To the liquor which has been run down into the second dissolver, a quantity of roasted ore is again added to neutralize the bisulphate of soda. As soon as this has taken place the liquid settles very rapidly to a clear solution, which is pumped through a clarifying filter press, and is then ready to have its zinc deposited in the electrolytic vats. At the Standard Co.'s plant only one electrolytic vat was used, as the dynamo power at disposal there was limited. Nine anodes were used and eight cathodes of the largest size each with an area of 8 sq. ft. What can be done with one vat can be done with any number on a large scale, a reduplication of vats of the same sizes being all that is necessary. The plant was well designed and erected and the results of running a large quantity of ore gave a remarkable good current consumption. Although 4000 kw.-hr. were allowed in costing for the deposition of 1 ton of zinc, only 3000 kw. were actually required, the lowest reading over a period of 48 hr. having been 2680 kw.-hr. On another occasion a 70-hr. run gave 2860 kw.-hr. As the zinc is deposited from solution in the electrolytic vat, the bisulphate of soda with which it was combined is regenerated and is used again in the next dissolving. The working costs vary with the ore, but they are now well defined and it is possible to give with some accuracy an idea of the profit which the district will derive from the working of this process near Nelson.

"The plant is expensive, but very much less than that required for a zinc smeltery of similar capacity. To deal with all the ores in this district some millions of dollars would be required for the plant alone. Ores of almost any grade above 10 per cent zinc can be treated equally well, but naturally the cost of treatment for an ore with a larger zinc content is less. As an example, an ore containing 24 per cent zinc, 24 per cent lead and 20 oz. silver per ton may be taken. Almost any of the mines can supply this or a better grade by picking or slight concentration without much loss of value. Taking zinc at 5 cents a pound, the lowest price at which it is possible for an ordinary zinc smeltery to work, and allowing freight on metallic zinc as far as Europe or China; lead at £20 per ton (London), and silver at 50 cents per ounce, and deducting ample costs for treatment, depreciation, management and royalty, the miner would receive for his ore about \$18.50 net per ton. It is only necessary to deduct cost of mining, picking the ore and freight to determine the net amount per ton the miner would receive for his ore, i. e., in the

neighborhood of \$15 per ton. For a supply of 500 tons of ore per day the net benefit to the miners would amount to upward of \$2,500,000 per annum."

New Chloridizing Mill for Tintic District, Utah

By early spring next year it is expected that a new mill will be operating in the Tintic district, Utah, using the process of chloridizing, roasting and leaching that has been described on several occasions in this journal. The new mill is being built by a company representing a consolidation of interests formerly operating separately at Park City and Silver City, Utah. The Mines Operating Co. was treating ore from the Ontario mine at Park City, using the Holt-Dern roaster, while the Knight-Christensen Metallurgical Co. built a mill at Silver City and used a Christensen roaster. The Mines Operating Co. found it inadvisable to continue operations under the conditions existing at Park City, and the Knight-Christensen company lost its Silver City mill by fire. As a result of a combination of circumstances, a consolidation of interests has been effected, and the Tintic Milling Co. has been organized to construct a new mill on the site of the old Knight smelter at Silver City. The officers of the new concern are: Jesse Knight, president; W. Lester Magnum, secretary; George H. Dern, vice-president and general manager; Theo. P. Holt, superintendent. The new mill will contain three Holt-Dern and one Christensen roasters, and as the two will be operated under identical conditions, an excellent opportunity will be afforded to compare their relative merits. The Tintic district affords an unlimited quantity of ore suitable to the process, and the new venture is expected to prove a boon to the district and yield a satisfactory profit to its promoters.

The Iron and Steel Market

The runaway in steel prices has proceeded farther and has perhaps reached the stage at which buying for ordinary purchases ceases, for the time being at least. Buyers, as a rule, are covered for many months ahead and chiefly, of course, at prices much below the advanced level now ruling. New wants arise, but there will naturally be hesitation about filling them. Thus the Pennsylvania Railroad has decided not to buy the 11,000 freight cars against which it recently issued inquiries, announcing that bids made are too high and deliveries proposed too slow. In the case of ship material buyers might not hesitate to pay prices asked, or even higher prices, but there is not much ship material coming up now, the yards being already well filled with business.

A decided increase is noted in inquiry for structural material, suggesting that much more building is planned for next year than has occurred in the past two years, but it remains to be seen whether a large volume of business can be put through at present prices.

Specifications on steel contracts continue to come in at a great rate and the mills are falling still farther behind in deliveries. Some of these current contracts for bars, plates and shapes are at 1.15 cent and 1.20 cent and specifications now filed against them will not be filled in many instances in less than from two to four months, so that a very wide range exists in invoice prices, since the current quotation on bars, plates and shapes for shipment at mill convenience is 1.70 cent, and small lots of bars and plates are bringing as high as 2.00 cent.

From some quarters come urgent requests to the mills for quicker deliveries, but there is no doubt that many buyers are comfortably fixed as to deliveries, if, indeed, they are not receiving material faster than they need it. Naturally buyers take all the material their

low-priced contracts permit, even if this causes the accumulation of considerable stocks. A condition that always obtains after a material rise in steel prices is that of some buyers, who were fortunate in placing large contracts at low prices, accumulating stocks of material or piling up specifications at the mills, and the existence of such a condition makes it difficult to judge the future, for one does not know whether buyers would be as enthusiastic about securing deliveries if they saw the market become stationary or start declining. Their physical needs might be the same, but their interpretation of them would be different.

In most finished lines the mills have business on books insuring operation at full capacity to about the middle of next year, and the chief exceptions to this condition are lines that do not customarily book business so far ahead. As a rule, new contracts could be taken only for third quarter of next year, and the mills do not wish to look so far ahead.

The production of steel is not increasing further, as the limit of capacity had already been reached. It seems safe to estimate that ingots are being produced at the rate of more than 40,000,000 tons a year.

Pig Iron

Prices have been advancing sharply in the past fortnight in nearly all markets. The advance is due much less to heavy market demand than it is to the productive ranks being well obligated to buyers already. Until but a few weeks ago advances in pig iron were either forestalled, or were greatly restricted in amount by the blowing in of idle furnaces. With considerable suddenness the limit is being reached, there being few idle merchant furnaces remaining, and some of those cannot blow in because they are not provided with ore, and nothing can be done with them until ore comes down the lakes next season. There has been, on the whole, a slight improvement in pig iron demand from foundries, but the foundries are still far from taking the tonnages they require in normally prosperous times. The steel works are not heavy buyers, but such purchases as are made count strongly in the market for the reason that the steel works that normally buy iron had already bought for forward delivery and present purchases are of additional lots. The furnaces claim to be well sold up, and effect price advances without any difficulty, but experience shows that in conditions like those of the present furnaces may have the appearance of being more sold up than is the case. If an accident occurs in the market it always turns out that there was more unsold iron than was estimated. As the market is trending now there will be sharper advances in the next two months than have occurred lately, even though during November the average price of all important descriptions of pig iron has advanced fully \$1.50. The more important advances in the past fortnight are 50 cents in Bessemer and basic \$1 in found iron at valley furnaces, \$1 in the Buffalo market and 50 cents at Philadelphia and Cleveland. We quote: No. 2 foundry iron, delivered Philadelphia, \$17.75 to \$18.25; f.o.b. furnace, Buffalo, \$17.50 to \$18; delivered Cleveland, \$17.30; f.o.b. furnace, Chicago, \$17.50 to \$18; f.o.b. Birmingham, \$13.50 to \$14; at valley furnaces, 95 cents higher delivered Pittsburgh; Bessemer, \$17.50 to \$18.00; basic, \$16.50 to \$17; No. 2 foundry and malleable, \$17; gray forge, \$16.50.

Statistics returned from upper lake docks indicate that the movement of iron ore out of the Lake Superior region this season, including all-rail traffic, will total about 47,000,000 tons, or within 3,000,000 tons of the record, made in 1913. Ore prices for next season are expected to be announced shortly, at an advance of 75 cents to \$1 a ton over the schedule for the season just

closing, a schedule identical with that of 1912 and 1914, prices in these three years having been the lowest since 1904.

Steel

Transactions in billets and sheet bars have been limited almost entirely to sales to regular consumers, who depend upon some particular source of supply, and odd lots disposed of by brokers. There is no regular market for Bessemer steel, while there is practically no market at all for open-hearth. Prices nominally representing the market so far as there can be said to be one, are approximately as follows: Bessemer billets, \$28; sheet bars, \$28 to \$28.50; open-hearth billets, \$29 to \$30; sheet bars, \$29.50 to \$30, f.o.b. maker's mill, Pittsburgh or Youngstown. Forging billets are \$48 to \$50 and wire rods about \$35.

Finished Steel

Prices have been advancing sharply almost all along the line. Nov. 18 the Carnegie Steel Company advanced bars, plates and shapes \$2 a ton, to 1.70 cent, this price instantly becoming the minimum of the market. Premiums are the rule in bars and plates for early deliveries. Prices quoted are f.o.b. Pittsburgh unless otherwise noted, and are for ordinary deliveries:

Rails, standard sections, 1.25 cent for Bessemer, 1.34 cent for open-hearth, f.o.b. mill, except Colorado.

Plates, tank quality, 1.70 cent.

Shapes, 1.70 cent.

Steel bars and bands, 1.70 cent, base; steel hoops, 1.75 cent, base.

Iron bars, 1.759 cent to 1.809 cent, Philadelphia; 1.60 cent, Chicago; 1.60 cent, Pittsburgh.

Plain wire, 1.75 cent; wire nails, \$1.90 per keg, base; painted barb wire, 2.05 cent; galvanized barb wire, 2.75 cent.

Sheets, blue annealed, 2.00 cent to 2.10 cent; black, 2.40 cent to 2.50 cent; galvanized, 4.50 cent to 5.25 cent; painted corrugated, 2.20 cent to 2.30 cent; galvanized corrugated, 4.55 cent to 5.30 cent.

Tin plate, \$3.60 for 100-pound cokes.

Steel pipe, $\frac{3}{4}$ to 3-in., black, 78 per cent off list; galvanized, 63 $\frac{1}{2}$ per cent off list.

Boiler tubes (less than carloads) 3 $\frac{1}{2}$ to 4 $\frac{1}{2}$ in., 69 per cent off list.

Structural rivets, 2.40 cent; boiler rivets, 2.50 cent.

Non-Ferrous Metal Market

The situation in this market for the two weeks ending Nov. 23 has been one of advancing tendency in all directions. These advances for the most part have been legitimate, and due to increased demand. There is great difficulty in obtaining spot deliveries and futures are well sold up.

Copper.—On Nov. 8 electrolytic was quoted at 18@18 $\frac{1}{4}$ cents. This price remained fairly steady for a week despite the closing of the Nichols Refinery at Laurel Hill, L. I., by strike on Nov. 10, which removed more than a million pounds from the daily production. On Nov. 12 the price had risen to 18 $\frac{1}{2}$ cents for electrolytic, and by the 17th the price was 19 cents. Consumers may have been frightened into buying by the advancing market. By Nov. 20 the price had reached 19 $\frac{1}{2}$ cents, and on the 22nd the much talked of 20-cent copper was realized. Exports up to Nov. 23 were 16,570 tons (compiled by N. Y. Metal Exchange). The strike at Laurel Hill was settled Nov. 16.

Tin.—The condition in the tin market has been one of extreme irregularity. The limited supply available and the uncertain reports have upset the market. Demand has not been normal, otherwise an acute situation

might have developed. On Nov. 10 the spot market was 37 cents, when the report of the closing of the Suez Canal drove the price up until 44 $\frac{1}{2}$ cents had been reached on Nov. 15. The market then reacted when it was found that this report was not true, and on the 22nd the market was weak and dull, with spot at 39 cents. Total arrivals up to Nov. 23, 1100 tons.

Lead.—The lead market was strong, with two advances recorded by the Trust between Nov. 8 and 23. On Nov. 10 the price was advanced 0.15 to 5.15 and on Nov. 15 it was advanced 0.10 to 5.25. This price has held steady. The market is pretty well in the hands of the Trust, and there was little inquiry in the outside market. There has been an absence of speculation.

Spelter.—The demand for this metal has been extremely good, and an advance of substantially 3 cents was recorded. On Nov. 8 spot was quoted at 15.20 New York. During the next two weeks heavy buying and a flood of inquiry sent prices soaring, and on Nov. 22 prime virgin spelter was quoted at 18.75@19.25 on the N. Y. Metal Exchange.

Other Metals.—Aluminium has held firm at approximately 56@59. On Nov. 18 sales of 200 tons were reported at close to 58, which staggered the market momentarily, but the price has, however, held firm. Antimony has advanced about 3 cents per pound on heavy demand, and on Nov. 22 was quoted at 38@39. Silver rose from 50 $\frac{1}{8}$ on Nov. 8 to 52 $\frac{1}{8}$ on Nov. 22. Platinum, on account of the French embargo, is now quoted \$68 for soft. Quicksilver rose \$5, to \$105 per flask.

The Roessler & Hasslacher Chemical Co., New York, was awarded a gold medal at the Panama-Pacific Exposition for its exhibit of cyanides.

Iron in Utah.—Articles of incorporation have been filed by the Utah Iron & Steel Co., capitalized at \$300,000. This is the first step in developing the iron fields of Utah, the company having a new plant in operation at Midvale which may be enlarged. The headquarters will be at Salt Lake City.

Antimony mining in Japan has had a great impetus as a result of the war. Prior to 1914 the mining of this metal in Japan was neglected on account of the cheapness of imported Chinese sulphide of antimony, but at the present time the production is about 50 tons a month. Consumption in Japan is roughly about 450 tons a month.

Mineral Resources of Quebec.—According to a recent consular report, by far the greater part of the Province of Quebec is as yet unprospected. The mineral products produced are aluminium, asbestos, antimony, arsenic, chromite, copper, feldspar, gold, graphite, iron, lead, zinc, magnesite, mica, molybdenum, phosphate and other. An important amendment to the mining laws of this province was passed in 1914. Lands containing combustible natural gas, mineral oil, or naphtha may now be staked, either ordinary or for a long term, upon conditions which are favorable for exploration.

The famous "silver sidewalk," considered to be the largest slab of native silver in the world, has been removed from the La Rose property at Cobalt, Canada, and placed in the museum in Toronto. The preservation of this remarkable specimen of silver ore was undertaken by the provincial government, and the greatest care was exercised in its removal. It is reported that drill holes surrounding the mass were filled with unslaked lime, after which water was added and the holes plugged. The pressure exerted by the slaking of the lime dislodged the slab with the least possibility of fracture.

Alumina in Steel

BY GEORGE F. COMSTOCK

One reason for the value of the microscope in the examination of steel is its ability to give evidence as to the cleanness of the metal, or as to the number and character of the non-metallic inclusions embedded in it. By preparing a surface with a sufficiently perfect polish, not only the number of inclusions per given area can be ascertained, but their shape, color and arrangement through the metal can also be plainly seen if lenses of good quality and fairly high magnifying power are used. After examination of many samples of steel with especial regard to the non-metallic inclusions, experience is acquired by which these inclusions may be classified into several different types based on their appearance in a well-polished unetched steel surface.

The sulphides are the most common inclusions in steel, and their appearance is well known to all metallographers. Inclusions of this type used to be known as "manganese sulphide," but investigations made in recent years have shown that iron sulphide also enters into their composition to a considerable and variable extent. Next to the sulphides, the most common inclusions are the silicates, or true slag inclusions. These are always darker than the sulphides and usually of more irregular form. To some writers, all inclusions except sulphides are "slag," and others do not even make this exception, calling everything "slag" in a polished section that is not metal. From evidence obtained recently in these laboratories, however, it has seemed justifiable to differentiate between slag or silicate inclusions and those consisting wholly or chiefly of titanium nitride, or of alumina, respectively. The titanium nitride inclusions were described in the METALLURGICAL AND CHEMICAL ENGINEERING for September, 1914, Vol. XII, page 577, and are easily distinguished from all other inclusions in steel by their pink

color. In this report is presented the evidence on which the identification of alumina in steel has been made.

A few years ago an ingot of good steel was treated, in the mold, with a large excess of aluminium, and the bars forged from this ingot were very seamy. Photomicrograph Fig. 1 shows the appearance of the inclusions in this steel, which must evidently be alumina, since nothing but this was added to the molten metal.

These inclusions are in the form of small rounded spots, arranged close together in an elongated streak. They are of a very dark bluish-gray color, when examined with the white light of an electric arc, appearing black unless highly magnified, and it is practically impossible to polish them without forming little pits around each inclusion. If the polishing is done very carefully these pits may be kept very small; but with certain methods of polishing the pits are made so large that the original inclusions cannot be seen at all. Titanium nitride inclusions will form pits in the same way if poorly polished. If the specimen is not rotated constantly during the final polishing, the pits take the form of short scratches, and each inclusion will have a little tail, like a comet.

It will be noticed in photomicrograph* Fig. 1 that although this shows a longitudinal view of a bar, the individual inclusions have not been elongated by the forging at all, but merely the group as a whole has been drawn out into a streak. Compare this with Fig. 2, which shows some silicates in the web of a rail, and a great difference will be apparent, for the silicates have been elongated even by the crosswise pressure of the rolls forming the web, while the drawing effect of the forging did not elongate the alumina particles even in the same direction that the bar was drawn out.

The difference between inclusions of alumina and ordinary slag or silicates in steel may then be summarized as follows:

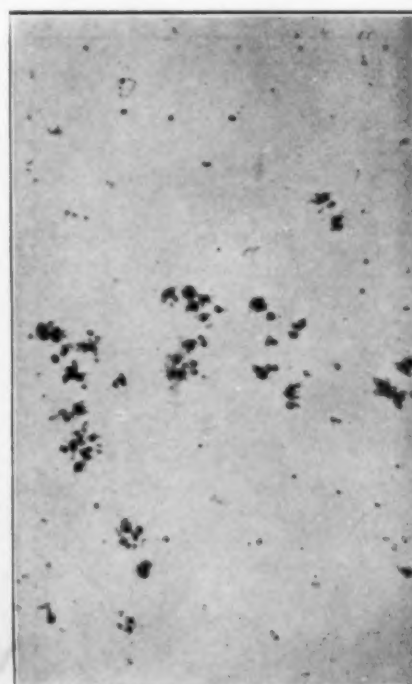
*All photomicrographs show unetched sections.



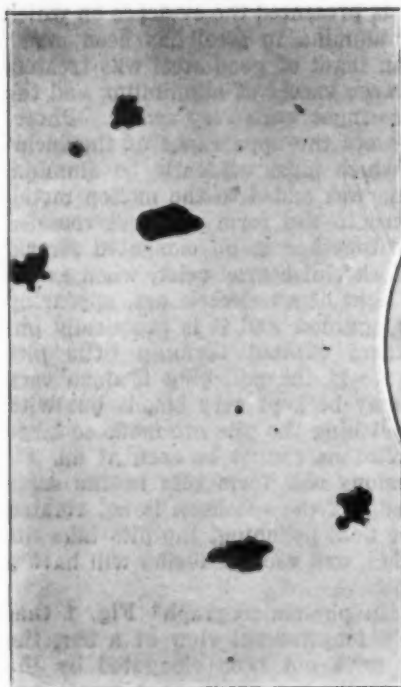
NO. 1—INCLUSIONS OF ALUMINA IN SECTION PARALLEL TO DIRECTION OF FORGING OF STEEL TREATED WITH A LARGE EXCESS OF ALUMINIUM. THE FEW GRAY SPOTS ARE SULPHIDES. MAGNIFIED 200 DIAMETERS AND UNETCHED.



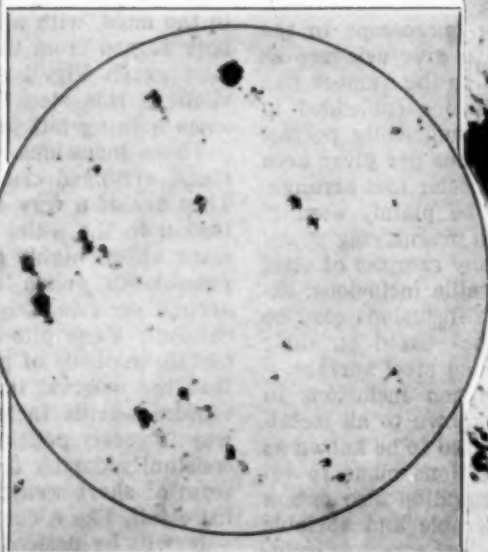
NO. 2—SILICATE OR SLAG INCLUSIONS (WITH A FEW GRAY SULPHIDES) IN THE CROSS SECTION OF THE WEB OF A RAIL. MAGNIFIED 200 DIAMETERS AND UNETCHED.



NO. 3—ALUMINA INCLUSIONS IN A SOFT STEEL INGOT IN WHICH MORE ALUMINA WAS FOUND BY CHEMICAL ANALYSIS THAN ALL OTHER OXIDES. MAGNIFIED 200 DIAMETERS AND UNETCHED.



NO. 4—SILICATE OR SLAG INCLUSIONS IN THE CROSS SECTION OF THE HEAD OF A RAIL. MAGNIFIED 200 DIAMETERS AND UNETCHED.



NO. 5—SECTION OF BROKEN SPLICE-BAR IN WHICH 0.019 PER CENT OF ALUMINA WAS FOUND BY CHEMICAL ANALYSIS. 200 DIAMETERS.



NO. 6—ALLOY, MADE IN SMALL ELECTRIC FURNACE, OF ALUMINO-THERMIC FERRO-TITANIUM WITH SWEDISH IRON. 400 DIAMETERS.

(1) Silicate inclusions will generally take a fairly smooth polish in a section prepared for microscopic examination, while alumina is very hard to polish without pitting.

(2) Silicate inclusions are always elongated in the direction of rolling or forging, while alumina particles

are not (the *groups* of particles are, of course, elongated, but not the particles themselves).

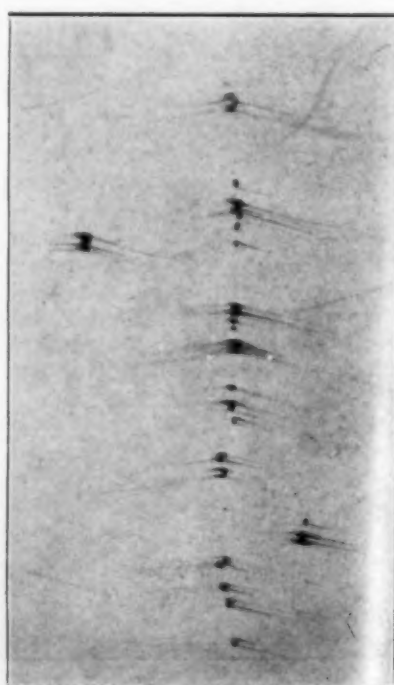
(3) Silicate inclusions are often found of quite large size (as well as very small), while particles of alumina are always small, and do not seem to coalesce into larger bodies even when closely grouped together.



NO. 7—GROUP OF SEGREGATED INCLUSIONS IN CROSS SECTION OF THE WEB OF A RAIL, IN WHICH 0.003 PER CENT OF ALUMINA WAS FOUND BY CHEMICAL ANALYSIS. MAGNIFIED 200 DIAMETERS.



NO. 8—AVERAGE VIEW OF THE CROSS SECTION OF THE HEAD OF A RAIL IN WHICH 0.010 PER CENT OF ALUMINA WAS FOUND BY CHEMICAL ANALYSIS. MAGNIFIED 200 DIAMETERS.



NO. 9—STREAK OF ALUMINA INCLUSIONS BADLY POLISHED, SHOWING PITS AND SCRATCHES. MAGNIFIED 200 DIAMETERS AND UNETCHED, SHOWING "COMET TAILS" DUE TO POLISHING.



NO. 10—SAME STREAK SHOWN IN NO. 9 AFTER GRINDING AND MORE CAREFUL POLISHING. MAGNIFIED 200 DIAMETERS AND UNETCHED. THE ALUMINA INCLUSIONS SHOW NO "COMET TAILS" AS PITTING HAS BEEN AVOIDED.



NO. 11—LONGITUDINAL SECTION OF A STREAK THAT CAUSED THE TOP OF THE HEAD OF A RAIL TO SHELL OFF, SHOWING ALUMINA INCLUSIONS AT THE LEFT AND SULPHIDES AT THE RIGHT. MAGNIFIED 200 DIAMETERS AND UNETCHED.



NO. 12—TYPICAL LARGE SILICATE OR SLAG STREAK IN THE CROSS SECTION OF THE WEB OF A RAIL, SHOWING ALSO TWO PARTICLES OF ALUMINA AND SOME FINE GRAY SULPHIDES. MAGNIFIED 200 DIAMETERS AND UNETCHED.

These characteristics of the alumina inclusions agree with what is known of the properties of alumina. Its great hardness and brittleness would account for the pitting effect; its infusibility would account for the

small size of the particles and the tendency not to coalesce; and both of these properties together would account for the particles not being elongated by forging or rolling of the steel in which they are embedded.



NO. 13—ALUMINA INCLUSIONS IN CLEAN RAIL STEEL TREATED WITH AN EXCESS OF ALUMINIUM. MAGNIFIED 200 DIAMETERS.



NO. 14—SLAG INCLUSIONS IN SIMILAR STEEL TREATED WITH OXIDE OF TITANIUM BEFORE CASTING. MAGNIFIED 200 DIAMETERS.



NO. 15—STEEL TREATED WITH OXIDE OF CHROMIUM, SHOWING END OF A BROAD STREAK, WITH TYPICAL SMALL PARTICLES EMBEDDED IN THE METAL. 200 DIAMETERS.

Inclusions having all the characteristics of those seen in steel known to contain alumina have been found in many samples of steel examined in the course of our work here, but it could not be stated positively that any of these were alumina until a method was found for determining this substance quantitatively by chemical analysis. Eventually such a method was found, not only in our laboratory, but also independently by Mr. F. O. Kichline, chemist at the Saucon plant, Bethlehem Steel Co. (published in the September, 1915, issue of the *Journal of Industrial and Engineering Chemistry*) and the results from it have been very interesting.

All samples in which more than the merest trace of alumina was found by analysis were seen to contain the typical inclusions as described above, and those in which alumina was not found by analysis did not contain these inclusions. Furthermore, those in which more alumina was found by analysis contained more of these inclusions than those in which only a very little was found.

These facts have been considered as a good confirmation of the theory that the typical small inclusions, as described above, found in so many commercial steels are chiefly, if not wholly, alumina. Of course, the purity of the alumina in these inclusions is not known, but even if it is fluxed with some impurity it would seem proper to call the inclusions "alumina" when it is known that their character is determined by the presence of this substance.

Photomicrograph Fig. 3 shows the inclusions in an ingot of soft steel in which chemical analysis showed the presence of a much larger amount of alumina than of all other insoluble oxides, thus proving the presence of a considerable quantity of free alumina.

Fig. 4 shows some ordinary slag inclusions in a cross-section of the head of a rail.

This photomicrograph and Figs. 2 and 12 are included to show the difference between silicates and alumina.

Fig. 5 shows a section of a splice-bar which broke in a railroad track and was found by analysis to contain 0.019 per cent of alumina.

Fig. 6 is an alloy made on a very small scale of Swedish iron and aluminothermic ferrotitanium, and shows alumina which was not present in the original iron.

Fig. 7 shows a spot where alumina particles are segregated in the web of a rail, in which 0.003 per cent was found, and Fig. 8 is an average view of the head of another rail containing 0.010 per cent of alumina. Fig. 8 shows the most common mode of occurrence of these particles, that is, scattered thinly through the metal. In Fig. 7 most of the specimen showed very few inclusions, but the small segregated streaks shown in the photomicrograph are more dangerous to the life of the rail than the larger total amount of alumina in Fig. 8, where it is arranged differently.

Figs. 9 and 10 are photomicrographs of the same streak, differently polished; the former shows the



NO. 16—STEEL TREATED WITH OXIDE OF NICKEL, SHOWING TYPICAL INCLUSIONS OF EITHER NICKEL OXIDE OR IRON OXIDE. MAGNIFIED 200 DIAMETERS.

comet-tails mentioned above, while in the latter the pitting has been more successfully avoided.

Fig. 11 is a view of some streaks that caused the top of the head of a rail to shell off from the rest of the section. The form of the alumina is well contrasted with the sulphides and with the large slag inclusion in Fig. 12, which is a large but typical silicate streak in the web of another rail.

All these photomicrographs show open-hearth steel, except possibly Fig. 5, which may be Bessemer, and Fig. 6, which is an electric-furnace melt. All show unetched surfaces, and all except Fig. 6 were taken with a magnification of 200 diameters. Fig. 6 was magnified twice as much as the others.

As a final check on the accuracy of the assumption that all inclusions in steel, having exactly the same characteristics as those shown in photomicrograph Fig. 1, are alumina, several experimental melts were

made of rail steel with the addition of various oxides. The steel used was seen by previous microscopic examination to be practically free from non-metallic inclusions except sulphides, and ten-pound ingots were cast from it each treated with one of the following oxides: alumina, titanium oxide, chromium oxide, and nickel oxide. The tops of these ingots were forged from the original 2-in. square section to 1-in. square, and longitudinal sections were cut and carefully polished for microscopic examination.

The treatment with alumina was not successful, as this treated bar appeared just as clean as the original steel, so this melt was made over again with the use of metallic aluminium. Characteristic inclusions of alumina were then found in the forged bar, as shown in photomicrograph Fig. 13.

Fig. 14 shows part of a small streak found near the top of the bar made with titanium oxide. The rest of this bar was clean, and the inclusions shown would ordinarily be classed as silicates or slag. They show a peculiar duplex composition and would not be mistaken for alumina.

The steel treated with chromium oxide had small smooth purplish spots scattered all through it, but they were especially segregated in some streaks at the top of the ingot. Some of these inclusions were angular and some rounded, and like alumina they were not elongated by the forging. Where they were not too thickly segregated, however, they took a good smooth polish easily like the silicates, and from this fact and their purplish color they may be easily distinguished from alumina.

The photomicrograph Fig. 15 shows the end of one of the segregated streaks, where small particles of this oxide seem to be breaking away and entering into the steel, and the difference between these and alumina is readily apparent. The steel around these inclusions was very readily stained or tarnished during the polishing, and this effect, though noticed also with the nickel oxide and sometimes with titanium nitride, has not

been seen by the writer around the streaks of alumina.

The last photomicrograph, Fig. 16, shows some of the steel treated with nickel oxide, but these inclusions look exactly like iron oxide, and it may be that the nickel oxide was reduced by the metallic iron present, forming metallic nickel and iron oxide. At any rate these inclusions are entirely different in appearance from alumina. They look very much like sulphides, but are a little darker, and stand out more in relief from the surrounding metal.

The evidence available is thus seen to point uniformly to the conclusion that it is proper to identify as alumina all inclusions, seen in polished steel surfaces, that have exactly the characteristics of the alumina inclusions shown in photomicrographs Figs. 1 and 13. At present no other substance is known to the writer which has the same appearance in a polished steel surface as alumina. Ordinary slag inclusions, as well as the oxides of titanium, chromium and nickel, have been shown to be very different. Of course, there is a possibility that some substance may be found at some future time that has identically the same appearance, but the writer sees no reason to expect such an event, and believes that the work described above renders the identification of alumina in steel with the microscope a matter of as much certainty as the similar identification of sulphides or silicates, which has long been considered reliable.

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Solution Stratification as an Aid in the Purification of Electrolytes*

BY FRANCIS R. PYNE

In the electrolytic refining of copper there are two attendant evils of special prominence. One is the tendency of the electrolyte to stratify or form layers of various compositions, and the other is the accumulation of impurities in the electrolyte.

The stratification is caused by the formation of a heavy solution at the surface of the anode owing to the solution of copper, and by the formation of a lighter solution at the cathode owing to the plating out of copper. Unless the degree of stratification is greatly impeded, the result will be a very unsatisfactory cathode deposit, which will be at best heavy and lumpy at the bottom, while mealy and of poor quality at the top. The greater part of the current will pass through the strata of highest concentration, causing frequent short circuits and uneven dissolving action at the anode, with all the attendant deposition troubles.

As a preventive, the electrolyte is circulated throughout the tanks at a rate which varies with local conditions at each plant. Too rapid a circulation is liable to carry slime, if present, to the cathode, fouling it and causing metal losses in silver and gold. Too slow a circulation does not sufficiently suppress the stratification. Where the current density is high the circulation must be more rapid than would be the case with a lower density.

To show the degree to which stratification will take place, the figures in Table I are taken from a paper by Mr. W. T. Burns on "The Great Falls Electrolytic Plant" (Trans. A. I. Mining Eng., August, 1913). This illustration is selected on account of the high current density in use at Great Falls, 36 amp. per square foot (400 per sq. m.), which is conducive to very marked stratification. Samples of the electrolyte were

taken between the center pair of electrodes at the following levels below the solution line: Zone 1, 6 in. (15 cm.); Zone 2, 17 in. (43 cm.); Zone 3, 28 in. (70 cm.); Zone 4, 39 in. (100 cm.).

The figures represent the average of 104 samples taken during seven consecutive days, and the results are reported in grams per liter.

TABLE I

	Zone	Sp. Gr.	Acid	Copper
Circulation at the rate of 6 gal. (23 liters) per minute.....				
	1	1.213	174	38.4
	2	1.216	172	39.2
	3	1.216	171	40.2
	4	1.228	153	49.9
Circulation at the rate of 4 gal. (15 liters) per minute.....				
	1	1.211	163	40.0
	2	1.214	161	41.1
	3	1.216	159	42.0
	4	1.263	139	67.8
Circulation at the rate of 2 gal. (7.6 liters) per minute.....				
	1	1.210	167	37.8
	2	1.213	165	39.0
	3	1.217	162	41.0
	4	1.265	146	66.4
Circulation shut off for seven hours.....				
	1	1.185	179	23.4
	2	1.210	164	38.0
	3	1.230	151	51.8
	4	1.255	138	65.5

The above figures clearly show the increase in stratification with decrease in circulation.

Regarding the accumulation of impurities in the electrolyte, these impurities are present in the anode, whence they pass into solution through the action of the current, or they may come from some outside source, for example, an addition agent. If allowed to accumulate in the electrolyte without restraint they may foul the cathode by being deposited thereon either electrically or mechanically. They may also increase the resistance of the electrolyte causing thereby a loss of power, or they may increase the specific gravity so that the settling of the slime is hindered, with a possible loss of slime in the cathode. In order to prevent these undesirable effects, the impurities in the electrolyte are not permitted to go above certain limits, and this is accomplished by withdrawing a certain amount of electrolyte from time to time for the removal therefrom of the undesirable constituents, and this removal must be done in such a manner that the valuable ingredients, such as copper and acid, shall not be wasted.

The customary method in use at the present time is to withdraw daily from the electrolyte a certain portion which is either worked for bluestone ($\text{CuSO}_4 \cdot 5\text{H}_2\text{O}$), or else passed through tanks containing insoluble anodes. It is with the latter method that this paper deals. The treatment with insoluble anodes removes a large part of the copper from the electrolyte, but requires a large power expenditure, the output per kilowatt-hour being one to two pounds (0.45 to 0.90 kg.) as against an output of eight pounds (3.63 kg.) in the regular refining tanks. Moreover the quality of the deposited copper is greatly inferior to the regular cathodes.

The solution from this first set of insoluble anode tanks is then passed through a second set, usually at a slower rate of circulation, and this second set of tanks removes the balance of the copper and practically all of the arsenic and antimony. The solution is then treated in any suitable manner for the elimination of such other impurities as may be desired.

From the above it is evident that could a solution be secured from the regular tanks sufficiently low in copper to avoid treatment in the first set of insoluble anode tanks, there would result a considerable saving in power, assuming that it contained practically an undiminished amount of impurities.

*A paper read at the San Francisco meeting of the American Electrochemical Society, Sept. 17, 1915.

Knowing the tendency of the electrolyte to stratify, an endeavor was made to utilize this principle as a means to this end. Accordingly a tank was constructed in which the circulation was so arranged that stratification would readily occur. The electrolyte entered the tank just above the slime and was drawn off through two outlets, one at the solution line, the other just above the slime line and opposite the inlet. (U. S. Patent No. 1,148,798, see Met. & Chem. Nov. 1, '15, Pg. 816.)

The accompanying figure shows a sectional elevation

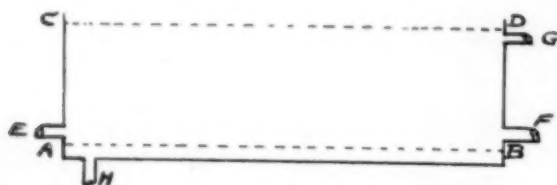


FIG. 1—SHOWING DIAGRAMMATICALLY THE TANK OPENINGS

of the tank as applied to copper refining. *AB* is the slime line, *H* the outlet for removal of slime from the tank, *CD* is the solution line. The electrodes have been omitted for the sake of clearness. The electrolyte enters the tank at *E* and is withdrawn through the upper and lower outlets, *G* and *F* respectively.

If the tank is filled with electrolyte and the upper outlet closed, any solution entering the tank through the inlet *E* will pass directly to the outlet *F* without disturbing the solution in the tank, providing the amount entering and leaving the tank is the same. If, however, the upper outlet *G* is open as well as the outlet *F*, a part of the incoming solution will flow through *F* while the remainder will rise in the tank to take the place of that withdrawn through the outlet *G*.

Now if the electrodes are in place and the normal current is passing and all circulation is stopped, the heavy solution formed at the anode by the copper passing into solution will stream downwards to the bottom of the tank, while the lighter solution at the cathode, due to the removal of the copper from solution, will rise. This is very clearly shown by the test made by Mr. Burns, in which the circulation was stopped for a period of 7 hr. Therefore, there will eventually be a stratum of heavy copper solution around the lower part of the cathode, while the solution in the upper part of the tank will be very low in copper and consequently lighter. The deposit on the lower part of the cathode will be rough and heavy while that on the upper part of the cathode will be loose and mealy.

As stated before, this is a very unsatisfactory condition of affairs, and it is therefore necessary to have such a circulation that the cathode will receive a satisfactory deposit throughout. This is accomplished by so regulating the flow from the upper and lower outlets that the heavy solution at the bottom of the tank will rise to such a height that there is a sufficient amount of copper at all points of the cathode. When proper regulation has been made, the regular tank room solution enters the tank through the inlet *E*, a solution high in copper is withdrawn from the tank through the outlet *F*, while the solution from *G* is low in copper and the cathode is in all respects similar to those drawn from the regular refining tanks.

The low grade solution withdrawn from the outlet *G* has been found in practice to be lower in copper than is usually obtained from the outlet of the first set of insoluble anode tanks mentioned above, and it is delivered directly to the second set of insoluble anode tanks for the removal of the last traces of copper.

The tanks have shown no increase in voltage over that of the regular refining tanks and there is no increase in the labor or material required in their operation. The solution withdrawn from the upper outlet can be varied through quite a range of copper content, and as it is a simple matter to equip a tank for this operation, a large or small number can be kept ready to respond to variations in the amount of solution to be withdrawn from the main circulation. They can be cut in or out at any time and in any part of the circuit, without reference to the age of the copper in the remaining tanks, which must be considered when using insoluble anode tanks unless the latter form a separate section apart from the regular refining tanks. A comparison of the two methods may be seen from the following:

Former Method.—Tank house electrolyte first passed through tanks containing insoluble anodes, to remove a portion of the copper, requiring high power consumption and giving a product of inferior quality, thence to a second set of insoluble anode tanks to remove the remainder of the copper and some of the impurities, after which the solution is treated in any desired manner.

Present Method.—Tank house electrolyte drawn directly from the special stratification tanks, requiring no increase in power and avoiding the production of inferior grades of product; thence to the final insoluble anode tanks, as above.

The net saving is therefore the operation of one set of insoluble anode tanks.

U. S. Metals Refining Company,
Chrome, N. J.

Determination of Mercury in Amalgamation Tailings

At the Buffalo mill, Cobalt, Ontario, Canada, high-grade silver ore is amalgamated in a tube-mill containing a large quantity of mercury and strong cyanide solution. The tailings contain mercury, both metallic and sulphide, which is recovered by dissolution in alkaline sulphide and precipitation with aluminum. The laboratory method of determining mercury in the products of the process is described by E. B. Thornhill in his paper presented to the A. I. M. E. To determine the total mercury (both metallic and sulphide) 0.5 g. of the material is well mixed with cast-iron filings, free from grease, and placed in a hard-glass tube, sealed at one end, with a contraction at about 2 in. from the sealed end. The mercury is distilled off, by heating the bulb containing the charge, and condensed in the tube just beyond the contraction. After the distillation is complete the contracted portion of the tube is heated, the bulb portion pulled off and the end of the tube sealed. The tube containing the condensed mercury is allowed to cool, filled one-half full of 0.1 per cent KCN solution, and 10 to 15 100-mg. gold beads added. Each bead will amalgamate with about 1 mg. of mercury. The tube is shaken until all the mercury is amalgamated; the beads are transferred to a small porcelain cup, washed with water, dried with alcohol and weighed. After retorting off the mercury they are again weighed. From ten to twelve determinations can be made in 1 hr. by this method and it is accurate enough for control of operations.

When it is necessary to know the proportionate amounts of mercury as metallic and sulphide two 2-g. samples are weighed out. One sample is digested with concentrated nitric acid and the other with a 10 per cent solution of sodium sulphide. The mercury in the residue from each is then determined by the method just given.

Flotation at the Consolidated Arizona Smelting Co., Humboldt, Arizona

(Editorial Correspondence)

The introduction of flotation, together with other changes in the milling scheme, at the copper concentrator of the Consolidated Arizona Smelting Co., increased the recovery of copper by more than 20 per cent over the best saving previously made with jigs, tables and vanners. The margin is quite sufficient to represent a difference between profit and loss, particularly when it is accompanied by a simplification of the process and reduction in cost for power and labor.

Milling ore from the company's Blue Bell and DeSoto mines is a silicious sulphide containing pyrite and chalcopyrite in a gangue of quartz and schist. The average composition of the ore and mill products is shown in Table I. In addition to mine ore, a variable quantity of mill tailing from former operations is also reground and concentrated. This material contains about 2 per cent copper.

TABLE I—AVERAGE COMPOSITION, ORE AND MILL PRODUCTS

(Gold and silver in oz. per ton. Other constituents in per cent)										
	Au	Ag	Cu	Insol	Fe	CaO	MgO	Al ₂ O ₃	S	
Milling ore	0.03	1.3	3.0	65.0	10.9	1.0	1.2	4.7	13.0	
Table Conc.	0.09	2.3	7.0	10.0	35.0	
Table Tailing	0.025	1.0	2.1	..	5.8	
Flotation Conc.	0.10	5.0	14.0	24.0	22.4	
Gen. Mill Conc.	0.09	3.04	9.52	16.5	30.0	
Mill Tailing	0.01	0.4	0.3	..	3.0	

During the month of September, 1915, the total quantity of ore milled was 7173 tons. The average grade of mine ore was 2.7 per cent copper, and of all material, including old tailing, 2.65 per cent. The percentage recovery of valuable metals was 90.9 per cent copper, 68 per cent gold and 72 per cent silver. The gross amount of copper in concentrates was 345,000 lb., of which 240,000 lb., or nearly 70 per cent, was recovered by flotation. The usual recovery by flotation will average about 75 per cent of the total. The ratio of concentration in the entire mill was about 4, and in flotation between 7 and 8.

Effect of Changes in Milling Practice

A diagram of the present flow-sheet is given in Fig. 1. The ore is crushed dry in Blake crusher and rolls, screened in trommels and handled by belt conveyors and elevators in the usual manner. Water is first added at the boot of the main elevator indicated at the right of the diagram. Hardinge mills are used for final grinding, and Wilfley tables and an 11-cell Minerals Separation flotation machine recover the valuable metals.

The changes that have been made during the past year and a half, under the direction of G. M. Colvocoresses, general manager, and J. N. D. Gray, mill superintendent, have been directed toward simplifying and increasing the efficiency of the process, eliminating useless machines and reducing the cost of operation.

By taking out the jigs several beneficial results were obtained. The cost for power and labor was reduced and the general grade of concentrate increased. Also more favorable conditions for flotation were created, because the jigs used a large amount of water which made it difficult to maintain that uniformity in the flow of mill pulp which is essential to good work in flotation. The abandonment of vanners was a natural consequence of the adoption of flotation.

In its present simplified form the milling scheme differs radically from that used in an earlier day. Harz jigs, tables and vanners were formerly used, but the

recovery of copper under that system did not exceed 70 per cent, in spite of the complicated treatment to which the ore was subjected. The pyrite, breaking coarser than the chalcopyrite, was easily concentrated, but the softer and more friable copper mineral entailed a heavy loss.

Similar ore conditions prevail to-day; but now the flotation process is effective at the very point in the treatment where tables and vanners proved inefficient. Wilfley tables are now used only for rough concentration, and flotation is relied upon for the major part of copper recovery. Reference to Table I will show that the proportionate iron and copper contents of table and flotation concentrates are reversed. Pyrite concentrates easily on the tables, but floats less readily than chalcopyrite. The flotation product, on the other hand, not only contains twice the percentage of copper found in the table concentrate, but also represents 75 per cent of the total recovery. Furthermore, about 60 per cent of the flotation product is finer than 200-mesh, showing the need of an efficient method of fine concentration if a heavy loss is to be avoided.

Data on Regrinding

Detailed figures on the performance of Hardinge mills are given in Tables II and III. These mills are lined with pebbles set in cement, but a Komata manganese-steel liner is to be given a trial soon. An ob-

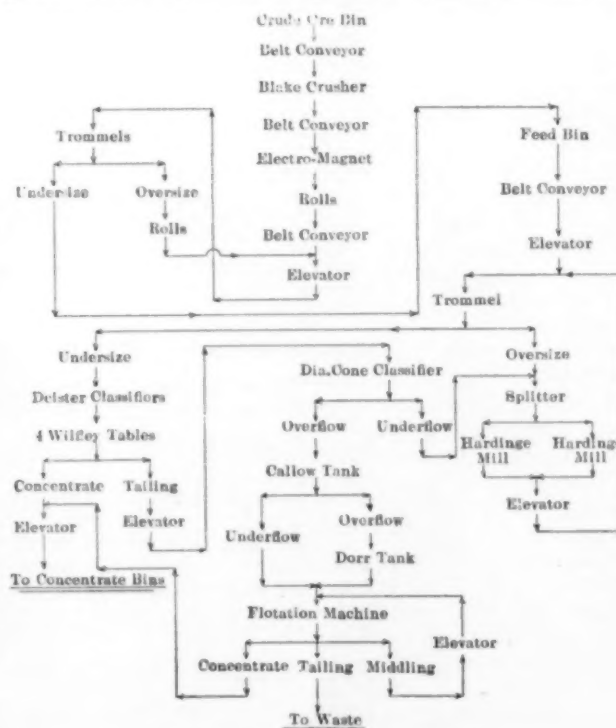


FIG. 1—CONCENTRATOR FLOW SHEET

jection to the present liner is the adverse effect on flotation results, caused by the cement which enters the pulp when a newly lined mill is put in operation. This objection will be overcome by the metal lining.

Selected pieces of copper-bearing rock have been used, instead of pebbles, as the grinding medium. The consumption is indicated in the tables under the headings "Feed to Mill Rock Tons," and "Per Ton Crude Ore." When the Komata lining is installed it is planned to conduct experiments with different grinders—selected ore, chrome-steel balls and "manganoid" balls.

Flotation of Table Tailing

The Consolidated Arizona Smelting Co. was one of the first in this country to adopt the Minerals Separation

TABLE II. SEPTEMBER HARDING MILL RECORD

Date	No. 1 MILL, BELT-DRIVEN [8 Ft. x 36 In. 27 R.P.M.]										No. 2 MILL, DIRECT-GEARED [8 Ft. x 36 In. 23 R.P.M.]									
	GrADINGS					Log					GrADINGS					Log				
	INLET					OUTLET					INLET					OUTLET				
	+	8	14	35	65	+	8	14	35	65	+	8	14	35	65	+	8	14	35	65
September 1	10.0	16.3	7.5	6.2	11.3	15.0	33.7	18.7	15.0	7.5	7.5	16.3	33.7	18.7	15.0	7.5	16.3	33.7	18.7	15.0
2	10.0	23.7	11.3	12.5	23.7	16.3	12.5	16.3	12.5	16.3	12.5	16.3	12.5	16.3	12.5	16.3	12.5	16.3	12.5	16.3
3	6.3	17.5	11.3	12.5	23.7	16.3	12.5	16.3	12.5	16.3	12.5	16.3	12.5	16.3	12.5	16.3	12.5	16.3	12.5	16.3
4	12.5	20.0	10.0	8.8	17.5	11.3	20.0	10.0	8.8	17.5	11.3	20.0	10.0	8.8	17.5	11.3	20.0	10.0	8.8	17.5
5	15.0	17.5	8.8	12.5	17.5	11.3	17.5	8.8	12.5	17.5	11.3	17.5	8.8	12.5	17.5	11.3	17.5	8.8	12.5	17.5
6	17.5	20.0	12.5	12.5	17.5	11.3	20.0	12.5	12.5	17.5	11.3	20.0	12.5	12.5	17.5	11.3	20.0	12.5	12.5	17.5
7	15.0	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5
8	15.0	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5
9	15.0	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5
10	15.0	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5
11	15.0	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5
12	15.0	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5
13	15.0	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5
14	15.0	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5
15	15.0	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5
16	15.0	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5
17	15.0	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5
18	15.0	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5
19	15.0	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5
20	15.0	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5
21	15.0	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5
22	15.0	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5
23	15.0	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5
24	15.0	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5
25	15.0	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5
26	15.0	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5
27	15.0	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5
28	15.0	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5
29	15.0	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5
30	15.0	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5	11.3	17.5	12.5	12.5	17.5
Total	14.9	22.6	10.3	11.7	15.9	9.9	14.7	100	3.6	14.3	14.7	26.7	40.7	100	34.67	74.6	4.12	68.4	22.35	93.4
Average	14.9	22.6	10.3	11.7	15.9	9.9	14.7	100	3.6	14.3	14.7	26.7	40.7	100	34.67	74.6	4.12	68.4	22.35	93.4
Per ton crude ore	14.9	22.6	10.3	11.7	15.9	9.9	14.7	100	3.6	14.3	14.7	26.7	40.7	100	34.67	74.6	4.12	68.4	22.35	93.4

TABLE III—GENERAL HARDING MILL RECORD

Date	No. 1 MILL										No. 2 MILL									
	GrADINGS					Log					GrADINGS					Log				
	INLET					OUTLET					INLET					OUTLET				
	+	8	14	35	65	+	8	14	35	65	+	8	14	35	65	+	8	14	35	65
January	15.2	19.2	10.8	20.6	17.0	17.2	100	15.4	24.2	12.4	14.1	33.9	100	37.99	61	124.33	73.3	22.31	18.1	20.5
February	13.8	19.3	12.3	21.3	16.9	16.4	100	15.4	24.2	12.4	14.1	33.9	100	37.99	61	124.33	73.3	22.31	18.1	20.5
March	13.4	22.8	12.4	16.0	17.3	18.1	100	8.9	21.0	14.0	19.1	38.1	100	38.29	67	124.66	78.4	19.40	16.5	20.0
April	15.6	23.0	10.9	17.7	19.7	13.1	100	11.7	28.0	13.8	20.2	36.3	100	36.66	72	129.14	59.0	22.52	14.8	20.0
May	19.0	26.2	12.7	14.5	14.3	13.3	100	6.6	19.4	14.4	23.2	36.6	100	36.66	72	129.14	59.0	22.52	14.8	20.0
June	15.6	21.5	11.2	16.3	19.0	16.4	100	7.1	21.1	15.7	22.6	34.4	100	35.30	75	126.95	61.2	22.8	15.2	21.4
July	9.6	21.4	12.9	19.1	19.7	17.3	100	8.6	22.0	14.5	20.5	34.4	100	35.30	75	126.95	61.2	22.8	15.2	21.4
August	9.1	21.1	12.0	18.9	22.7	16.2	100	6.7	21.8	15.9	21.0	34.6	100	35.21	74.3	116.35	66.5	21.60	15.0	23.9
September	14.9	22.6	10.3	11.7	15.9	9.9	100	3.6	14.3	14.7	26.0	40.7	100	34.67	74.6	123.57	66.8	22.51	15.0	26.4
Total	14.9	22.6	10.3	11.7	15.9	9.9	100	3.6	14.3	14.7	26.0	40.7	100	34.67	74.6	123.57	66.8	22.51	15.0	26.4
Average	14.9	22.6	10.3	11.7	15.9	9.9	100	3.6	14.3	14.7	26.0	40.7	100	34.67	74.6	123.57	66.8	22.51	15.0	26.4
Per ton crude ore	14.9	22.6	10.3	11.7	15.9	9.9	100	3.6	14.3	14.7	26.0	40.7	100	34.67	74.6	123.57	66.8	22.51	15.0	26.4

TABLE IV—SEPTEMBER FLOTATION RECORD

1915		DAILY										WEEKLY																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																											
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to pay the closest attention to the chemical condition of the ore. A splendid recovery is made on clean, bright, fresh sulphides, but when the minerals become tarnished or suffer surface oxidation to a slight extent, or if the gangue becomes oxidized and forms decomposition products such as oxide of iron and carbonate of magnesia, the flotation froth is almost immediately "killed" and a serious drop in percentage recovery is noted. Extreme care is used, therefore, to send to the mill only clean sulphide ore. Oxidized material from the upper levels of the mine is smelted directly or, if it is high in silica, it is used for converter flux and fettling.

The reduced recovery of January and June due to the involuntary treatment of oxidized material, is clearly shown in the flotation record, Table V, and in total mill recovery, Table VI.

TABLE VI.

Total Mill Recovery.		Per cent Copper.	
1915		Per cent Copper.	
January	74.6	74.6	
February	91.6	91.6	
March	90.2	90.2	
April	92.7	92.7	
May	92.7	92.7	
June	86.2	86.2	
July	92.2	92.2	
August	92.0	92.0	
September	90.9	90.9	

In Table V will be seen the variety of oils and mixtures tested with a view to getting the best metallurgical results with the cheapest materials. In July and subsequent months, "No. 200 oil" (refined wood creosote) has replaced cresol, turpentine and tar oil, with a consequent increase in stove oil, a cheaper oil. The mixture is now two parts stove oil and one part No. 200 oil.

Cost of Flotation

The total cost of concentration, including coarse crushing and flotation royalty, is slightly over \$1 per ton. In September it was \$1.03. The cost of the flotation process, exclusive of royalty, has been \$0.27 per ton over a period of six months. The oil cost has been gradually reduced to 2.8 cents per pound,¹ and the consumption averages a little less than one pound per ton of crude ore. The power consumed by the flotation machine for a period of 24 hours is 32.7 kw.

The record recovery in January made at this plant are but part of a comprehensive plan laid out by the management for placing the company's entire property—mines, concentrator and smelter—on an economic and profitable basis. The foregoing data on the mill speak for themselves and indicate the vast

¹Pounds per gallon: Stove oil, 7.12; wood creosote, 8.25; fuel oil, 7.48.

improvement that has already been made. With the anticipated development of the mines and increased mill capacity it is expected that the tonnage treated will be increased from 225 to 400 tons per day and that the cost for milling will be reduced from about \$1 to under \$0.90 per ton. In a later issue we shall publish some notes on the smelting practice at Humboldt.

To Mr. Colvocoresses and members of his staff we are indebted for many courtesies extended and for permission to publish valuable and timely data on the flotation of copper ores.

The Treatment of Sewage by Aeration in the Presence of Activated Sludge*

BY EDWARD BARTOW

Director of the Illinois State Water Survey

Sewage treatment by aeration in the presence of sludge is the latest development in sewage disposal. Air has always played an important rôle in sewage disposal. The earliest application of air was the exposure of sewage on the ground or in shallow pools. The disposal of sewage by irrigation is, therefore, an aeration process. No more sewage can be added to land than can be thoroughly oxidized. The disposal of sewage by dilution in streams also depends on the amount of air present. The amount of sewage which can be purified by a stream is limited by the amount of dissolved oxygen present. The efficiency of the stream depends upon the amount of water or the amount of oxygen in solution or the possibility of reaeration.

Intermittent sand filtration, where sewage is added intermittently to sand beds, is an aeration process, for between the periods of flooding with sewage, air is allowed to enter the pores of the sand. The action of contact beds is of a similar nature. Coarser material is used and between the periods of flooding, air enters the interstices and is the purifying agent. Sprinkling filters, the most practical process up to the time of the suggestion of activated sludge, depend upon aeration accomplished by spreading the sewage in a finely divided state into the air.

Preliminary to these aeration processes preparatory treatment is necessary. Preparatory treatment consists in using screens, grit chambers, settling tanks, digestion tanks or chemical precipitation. The preparatory treatment varies according to conditions. For example, three cities visited in Europe use different degrees of preparatory treatment prior to disposal by dilution. Munich uses no screening whatever, the water in the Isar being of sufficient quantity and sufficiently aerated to dispose of the sewage. Hamburg can dispose of its sewage in the lower Elbe using only coarse screening, but Dresden on the upper Elbe must pass its sewage through fine screens before emptying it into the river.

Grit chambers and settling tanks remove suspended matter. The amount of purification is comparable with the amount of purification by screening.

Digestion tanks accomplish the partial destruction of the suspended solids and soluble organic matter by anaerobic bacteria.

The addition of chemicals assists sedimentation and retards digestion, giving an increased amount of sludge, but a much improved effluent. Neither the screening nor the sedimentation nor the digestion nor the chemical precipitation produce complete purification. Aeration processes must complete the purification.

The latest process, the aeration of sewage in the

presence of sludge, has had a gradual development. Numerous experiments of blowing air into sewage have been made both in America and in Europe. Until recently none of the experiments were at all promising and the conclusion was that such means of purification was not practical. In this country, the first promising method was that used by Black and Phelps¹ in New York. They blew air through the sewage as it passed over a series of inclined wooden gratings. This sewage was in contact with the air for varying periods up to twenty-four hours. The results were promising enough to cause Black and Phelps to recommend the construction of a larger plant for Greater New York. That plant has not been constructed and the experimental tank has been adapted to experiments with activated sludge.

The next experiments are those reported by Clark and Gage and Adams² at the Massachusetts State Board of Health Experiment Station in Lawrence. Air blown through sewage reduced the organic constituents. The seeding of the sewage with green growths accelerated the action. Their best results were obtained when the tank contained slabs of slate covered with a brown growth of sewage matters. This treatment produced an effluent which could be filtered at several times the normal rate. It, however, simply prepared the sewage for addition to sand beds, and was not considered a final process.

Gilbert J. Fowler of the University of Manchester was in this country in November, 1912, in connection with the disposal of the sewage of Greater New York, and visited the Massachusetts Experiment Station. Fowler and Mumford carried on experiments with a specific bacillus, which they named M-7, which was collected from the waste water from a colliery. This bacillus with aeration has the power of separating iron as ferric hydroxide from iron-bearing sewage, carrying down with it the suspended matter and furnishing a non-putrescible effluent. Fowler suggested to Arden and Lockett, who were in charge of the Manchester Sewage Disposal Works, that they try experiments in aerating sewage on lines somewhat similar to what he saw in Massachusetts. As a result the *activated sludge process* is being developed.

DESCRIPTION OF ACTIVATED SLUDGE PROCESS

The first description of it was given by Arden and Lockett³ on April 3, 1914, at a meeting of the Manchester Section of the Society of Chemical Industry. In their first experiments Arden and Lockett used 5-pint bottles and drew the air through the sewage by means of an ordinary filter pump. Air was drawn through the sewage until it was completely nitrified, requiring about five weeks. The supernatant liquid was drawn off and additional sewage added. This method of treatment was repeated a number of times with the retention each time of the deposited solids. As the amount of deposited matter increased the time required for each succeeding oxidation gradually diminished. Finally a well-oxidized effluent, equal to that from efficient bacterial filters, was obtained in from six to nine hours.

In their second series of experiments, reported to the Manchester Section of the Society of Chemical Industry⁴ on Nov. 6, 1914, they used barrels of 50 gal. capacity and added the air through porous tile. They had tried treatment with a continuous flow of sewage without conclusive results. In later experiments they used tanks of 20,000 gal. capacity. Their results were very satis-

¹Report Concerning Location of Sewer Outlets and the Discharge of Sewage into New York Harbor (1911), 64-78.

²Annual Report Mass. State Bd. of Health (1913), 45, 285-304.

³Jour. Soc. Chem. Ind., vol. 33, p. 523-39.

⁴Jour. Soc. Chem. Ind., vol. 33, p. 1122-4 (1914).

*A paper read at the San Francisco meeting of the American Institute of Chemical Engineers, in the Palo Alto session, on Aug. 27, 1915.

factory and led to additional work in England, especially at Salford, where Duckworth⁵ and Melling⁶ adapted scrubbing filters to the use of the activated sludge process with great success.

In August, 1914, I had the privilege of meeting Professor Fowler and of seeing the work which had been done under his direction. On returning to this country, consulting with Fowler, experiments⁷ were begun with F. W. Mohlman at the University of Illinois on Nov. 2, 1914, using 3-gal. bottles. On Jan. 4, 1915, a tank 9 in. square and 4½ ft. deep was put in operation. In the bottom of this tank was placed a porous plate made of material as "Filtros,"⁸ furnished by the General Filtration Company, Rochester, N. Y.

The results of our first experiments were similar to those of Ardern and Lockett. Sewage placed in the bottles or in the little tank was submitted to a current of air for a sufficient period to oxidize it completely. The oxidation is best measured by the content of ammonia, nitrate and nitrite nitrogen. Oxidation has been carried to completion five different times with practically the same results. The time required for oxidation differed, but the courses of the reaction were similar.

As an example, in one of these experiments at the beginning 35 parts per million of ammonia nitrogen were present. (See Fig. 1.) The ammonia nitrogen remained practically constant for about four days, and then quite rapidly decreased, so that at the end of about seven days it was gone. There were no nitrates nor nitrites present in the raw sewage. The nitrites increased as the ammonia decreased. Then for a few

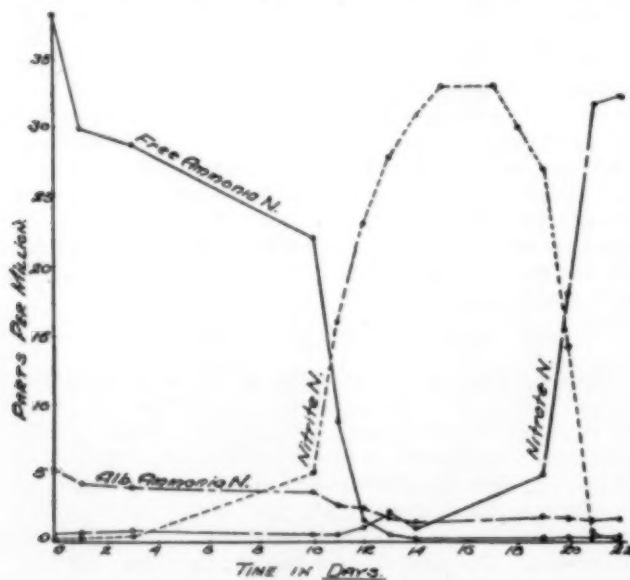


FIG. 1—NITRIFICATION OF SEWAGE—NO ACTIVATED SLUDGE PRESENT

days the nitrites remained constant and then decreased, the nitrates which were zero at the start increasing as the nitrites decreased. At the end of fifteen days nitrification was complete, the nitrite nitrogen had practically disappeared and the nitrate nitrogen had increased to about 25 parts per million. When the oxidation was complete the supernatant liquid was replaced by fresh sewage, the sludge being left. This process was repeated. With each change less time was

required for oxidation, as, for example, fifteen days for the first, four days for the second, and two days for the third, and so on, until with the thirty-first treatment oxidation was complete in five hours.

With accumulation of sludge as the process is repeated the reaction follows a different course; for example (see Fig. 2), using a sewage with 27 parts per million of ammonia nitrogen, the ammonia decreases and is practically eliminated in five hours. The nitrite nitrogen never increases to any extent; the nitrate nitrogen begins to increase almost at the start, increasing as the ammonia nitrogen decreases, and reaches its maximum when the ammonia nitrogen has disappeared. It is not necessary to obtain complete nitrification to obtain a clear or stable effluent. More information must be obtained before the amount of nitrification required can be known.

The process is undoubtedly bacteriological. The sludge is very rich in bacteria, but the number in the

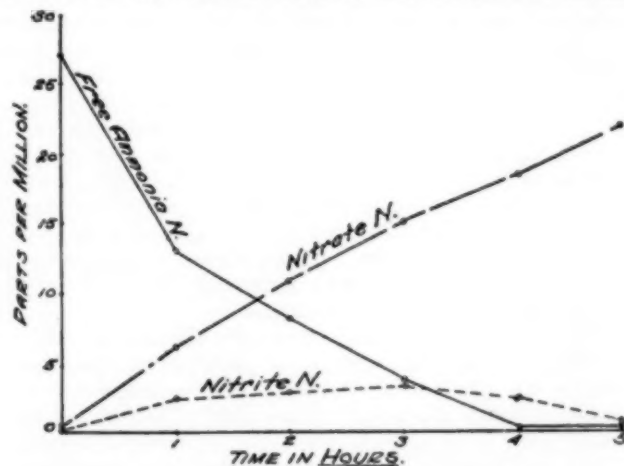


FIG. 2—NITRIFICATION OF SEWAGE—ACTIVATED SLUDGE PRESENT: 1 SLUDGE, 5 SEWAGE

effluent is comparatively small. Mr. Robbins Russell investigated the bacteriological properties of the sludge. Several species of bacteria were isolated from the sludge, including two kinds of nitrifiers and about fifteen other varieties. Experimenting with sterilized sewage it was found that the nitrifiers alone would not purify the sewage. Mixtures of the nitrifiers with the other bacteria would completely purify the sterilized sewage.

In an article which was presented before the American Chemical Society⁹ mention was made of some worms which developed in the process, and which were identified by Professor Frank Smith of the Zoological Department of the University of Illinois as *Aelosoma Hemprichi*. It seemed possible that these worms might play an important rôle in the purification process. Having isolated about 200 of these worms as completely as possible from the bacteria and sewage, Mr. Russell added them to sterilized ammonia broth and aerated it, obtaining practically no nitrification. The worms are, therefore, not essential and the bacteria which were isolated from the sludge did nitrify sterilized sewage.

PROPERTIES OF SLUDGE

The fresh sludge is odorless. It will putrefy if left with a large amount of water. After filter pressing it is stable. The dried sludge has an odor similar to that of fertilizers. The sludge is rich in nitrogen, 4.0 to 6.4 per cent. Its value as a fertilizer has been shown by pot cultures, made by W. D. Hatfield under the general direction of Prof. C. G. Hopkins and with the

⁵Surveyor, 46, 681-2 (1914).

⁶J. Soc. Chem. Ind., 33, 1124-30.

⁷J. Ind. Eng. Chem., 7, 318 (1915); Eng. News, 73, 647-8 (1915); Eng. Record, 71, 421-2 (1915); Eng. Contr., 43, 310-1 (1915).

⁸Through the courtesy of the manufacturers a number of the plates were exhibited at the meeting. The plates are made of a very pure and carefully graded quartz, fused together with powdered glass.

⁹J. Ind. Eng. Chem., 7, 318 (1915).

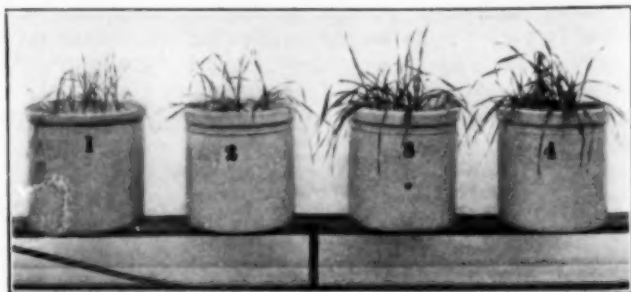


FIG. 3—POT CULTURES, WHEAT TWENTY-THREE DAYS AFTER PLANTING

Each pot contains pure white sand to which the same amount of plant foods except nitrogen have been added. No. 3 contains 20 g. dried activated sludge. No. 4 contains the same amount extracted with ligroin. No. 2 contains an equivalent of nitrogen from dried blood, and No. 1 contains no nitrogen.

assistance of Mr. J. C. Anderson of the University of Illinois. The pot cultures are made according to a standard method. Each pot contains 19,820 grams of pure white sea sand, 60 grams of dolomite, 6 grams of bone meal, and 3 grams of potassium sulphate; that is, each pot contained an equivalent of 5 tons per acre of dolomite, $\frac{1}{2}$ ton of bone meal, and 500 lb. of potassium sulphate. There is in the pots all of the constituents of a plant food except nitrogen.

Pot 1 contained no nitrogen. Pot 2 contained dried blood equivalent to 120 lb. of nitrogen per acre. Pots 3 and 4 contained the equivalent of 120 lb. nitrogen, or 1 ton of sludge per acre. Thirty wheat seeds were planted in each pot. In four days the plants were up in each pot, and in ten days were 5 in. high. At the end of eighteen days each pot was thinned to fifteen of the best plants. A noticeable difference was observed after twenty-three days. (See Fig. 3.) The pots fertilized with the sludge grew very much more rapidly than any of the others. The difference at the end of nine weeks (see Fig. 4) was very marked. The differences seemed too great to be true, so another series was planted. Similar results were obtained. (See Fig. 5).

Judgment of the results of such an experiment depend on the crop. In pot No. 1 the number of heads was fourteen and the number of seeds eighty-five, corresponding to 6.2 bu. per acre. In No. 2, fertilized with dried blood, the number of heads was fifteen, seeds 189,

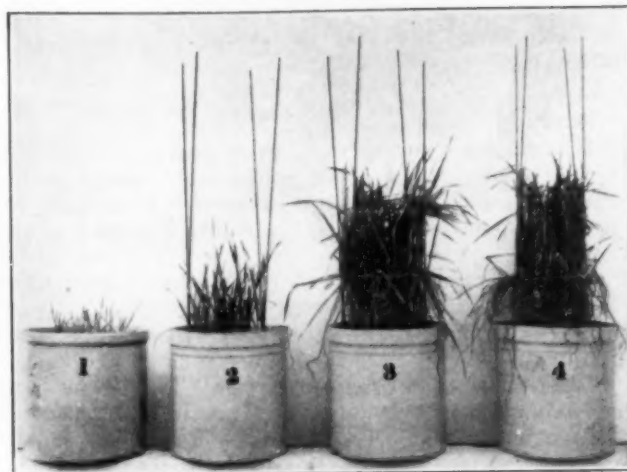


FIG. 4—POT CULTURES, WHEAT SIXTY-THREE DAYS AFTER PLANTING

Each pot contains pure white sand to which the same amount of plant foods except nitrogen have been added. No. 3 contains 20 g. dried activated sludge. No. 4 contains the same amount extracted with ligroin. No. 2 contains an equivalent of nitrogen from dried blood, and No. 1 contains no nitrogen.

corresponding to 13.6 bu. per acre. Dried blood is not a satisfactory fertilizer with some soils, which may account for poor results. In pots 3 and 4, fertilized with sludge, the number of heads was twenty-two and twenty-three, the number of seeds 491 and 518, corresponding to 35.9 and 37.7 bu. per acre. A very fair yield of wheat. The check experiment gave approximately the same results. The number of bushels per acre being 35.1 and 36.0.

The value of the sludge was tested with radishes and lettuce in the open field. Three plots, each 2 ft. by 3 ft., were laid out. One plot was fertilized with extracted sludge, a second with sludge and a third was not fertilized. Radishes and lettuce were planted in each. Thirty-eight days after planting the six best plants were taken from each plot. (See Fig. 6.) The difference in size was very marked. The fertilized lettuce weighed 40 per cent more than the unfertilized, and fertilized radishes weighed 150 per cent more than the unfertilized.

Both the pot cultures and the gardening experiments

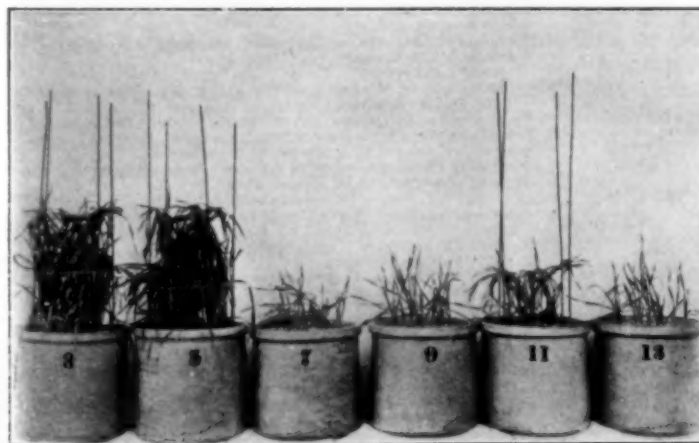


FIG. 5—POT CULTURES, WHEAT SERIES 2, THIRTY-FIVE DAYS AFTER PLANTING

Each pot contains pure white sand to which the same amount of plant foods except nitrogen have been added. No. 3 contains 20 g. dried activated sludge. No. 5 contains 20 g. of dried activated sludge from which other soluble matter has been removed. No. 7 contains the nitrogen equivalent from sodium nitrate. No. 9 contains the nitrogen equivalent from ammonium sulphates. No. 11 contains the nitrogen equivalent from gluten meal. No. 13 contains the nitrogen equivalent from dried blood.

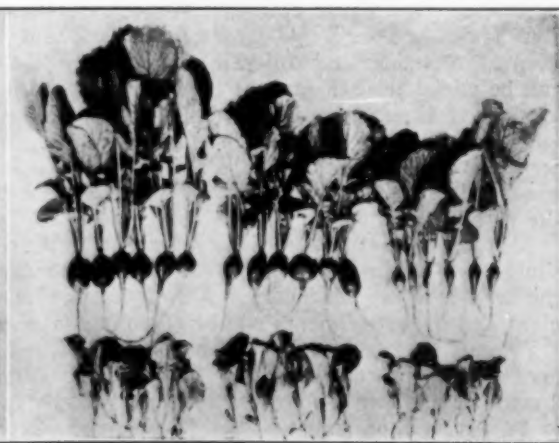


FIG. 6—GARDENING EXPERIMENTS, THIRTY-EIGHT DAYS AFTER PLANTING

Radishes. From left to right six radishes fertilized with extracted sludge, six radishes fertilized with sludge, six radishes unfertilized. Lettuce. Six heads fertilized with extracted sludge, six heads fertilized with sludge, six heads unfertilized. The fertilized lettuce weighed 40 per cent more than the unfertilized and fertilized radishes 150 per cent more than unfertilized.

show that the nitrogen in activated sludge is in a very available form, and that the sludge is valuable as a fertilizer.

APPLICATIONS OF THIS PROCESS IN AMERICA

The process is attracting a great deal of attention in America, and a very good statement¹⁰ concerning the work being done is given in *Engineering News*, July 15. Mr. M. N. Baker has given an editorial review of the subject.

The most extensive work is being done at Milwaukee. An article by Mr. T. Chalkley Hatton¹¹ gives a more complete account of the Milwaukee experiments. Two tanks of 1 x 5 x 10 ft. deep and one tank 10½ x 32 x 10 ft. deep have been operated on the fill-and-draw plan, and one tank, 10½ x 32 x 10 ft. deep, is operated on the continuous plan. The Milwaukee Sewerage Commission has awarded contracts for the construction of a plant to treat 2,000,000 gal. of sewage per day by continuous flow.

At Baltimore they have been working on a small scale, but have adapted two of the new Imhoff tanks for use by this process, and it is expected that in a short time they will be using activated sludge on a large scale.

At Washington, the Hygienic Laboratory of the Public Health Service is experimenting on a small scale and is co-operating with the department at Baltimore in their experiments.

At Cleveland experiments are being carried on in the sewage experiment station. They have adapted tanks 5 x 10 and 5 ft. deep, which they used in their sewage experiments to the process, and while it has barely begun they are getting promising results. Experiments are to be carried out on a larger scale.

At Regina, Saskatchewan, experiments on a considerable scale have been carried out and their results are reported by R. O. Wynne-Roberts.¹²

At Houston, Tex., they are planning to use the process in a plant to ultimately treat the sewage from 160,000 people. They do not expect to obtain complete nitrification, as they do not believe that a completely purified effluent is necessary.

In Chicago the Sanitary District of Chicago is using tanks about 2 ft. in diameter and 8 or 10 ft. high with quite satisfactory results, using the waste from the stockyards, one of the most difficult wastes that we have to treat.

At the University of Illinois, Urbana, we are operating four tanks on the fill-and-draw plan. Each tank has an area of 10 sq. ft. and a depth of 8 ft. About 400 gal. can be added to each tank at each filling. The bottoms are one-ninth, one-third, and all (in two tanks) covered with filter plates to try to determine the best arrangement for the diffusion of air. Of the three schemes tried one-third of the bottom with the rest slanting toward the plates seems better than the other schemes.

A study of methods of developing sludge shows that sludge can be developed from raw sewage by changing the sewage every six hours. A full description of the plant and the experiments will be given in a later article.

From 1 to 2 cu. ft. of air per gallon of sewage is needed, and the cost of the air, with electric current at 2 cents per kilowatt-hour, we estimate will be from \$5 to \$8 per million gallons. At Milwaukee, where they are using a little more air, they estimate the cost of the air with electric current, at 1 cent per kilowatt-hour, to be \$1 per hour of aeration. If they aerate for four hours that would be \$4 per million gallons for the cost of the air.

This method of sewage purification is giving such satisfactory results in the purification of sewage on a small scale under summer conditions, that it will be tried on a large scale and under winter conditions, so that we ought to know by spring whether it is a success.

State Water Survey, University of Illinois,
Urbana, Ill.

Columbia Section, American Institute of Mining Engineers

The fifth annual meeting of the Columbia Section, A. I. M. E., will be held in Wallace, Idaho, Nov. 19-20, 1915. Arrangements have been made for trips to the Tamarack, Interstate, Success and other mines and to the concentrating mills of the Cœur d'Alene district. The sessions of the first day will be devoted to the reading and discussion of the following papers: "Progress in Ore-Dressing," by D. W. Buckby; "Mine Accounting," by J. E. Chapman; "Zinc Smelting and Smelter Rates," by W. E. Henry. The business session and banquet will be held on the second day. A special invitation is extended to members of the Montana Section to attend this meeting.

Equipment of Arizona Smelters

In one of the papers presented at the recent sessions of the International Engineering Congress, in San Francisco, Dr. James Douglas gave the following condensed data on the equipment of smelting plants in the State of Arizona. A notable feature is the increasing number of reverberatory furnaces and the absence of blast furnaces in the newer plants.

- Copper Queen Cons. Mining Co. at Douglas, Arizona.
 - 16 blast furnaces, 18 ft. and 20 ft. by 42 ft.
 - 3 reverberatories, 91½ ft. by 19 ft.
 - 7 12-ft. converters.
 - 16 roasters.
- Calumet & Arizona Mining Co. at Douglas, Arizona.
 - 2 40-ft. blast furnaces.
 - 4 reverberatories, 19 ft. by 100 ft.
 - 6 stands 12-ft. upright converters of the Great Falls type.
 - 12 roasters.
- United Verde Copper Co. at Jerome, Arizona.
 - 4 blast furnaces, 27½ ft. by 4 ft.
 - 4 reverberatories, 100 ft. by 19 ft.
 - 6 roasters.
 - 5 12-ft. upright converters of Great Falls type.
- Arizona Copper Co. at Clifton, Arizona.
 - No blast furnaces.
 - 4 reverberatories, 19 ft. by 100 ft.
 - 6 roasters.
 - 6 stands 12-ft. upright converters of Great Falls type.
- The American Smelting & Refining Co. at Hayden, Arizona.
 - No blast furnaces.
 - 2 reverberatories, 19 ft. by 112 ft.
 - 8 roasters.
 - 3 converters.
- Detroit Copper Mining Co. of Arizona, Morenci, Arizona.
 - 1 large blast furnace.
 - 4 small converters.
- Old Dominion Copper Mining and Smelting Co., Globe, Arizona.
 - The Old Dominion Company has not yet added a reverberatory to its cupola plant, as the volume of concentrates from the small concentrator has been insufficient to economically feed one. Its plant consists of
 - 6 cupola furnaces, of which more than 4 are seldom used.
 - 2 12-ft. converters of Great Falls type.
 - one only of which is retained in blast, and it has, without relining, turning out 43,000,000 lb. of copper.
- Shannon Copper Co. at Clifton, Arizona.
 - 3 blast furnaces.
 - 2 stands of converters.
- International Smelting & Refining Co., Globe, Arizona.
 - (to treat the Inspiration and Miami Concentrates)
 - No blast furnaces.
 - 3 reverberatories.
 - 6 roaster furnaces.
 - 6 converters.
- Consolidated Arizona Smelting Co., Humboldt, Arizona.
 - (Humboldt Smelter)
 - 2 reverberatories, 19 ft. by 60 ft. and 19 ft. by 100 ft.
 - 1 Wedge roaster.
 - 2 converters.

Pulverizer Mill—The Fuller-Lehigh pulverizer mill is completely described in an excellent catalog recently issued by the makers, the Lehigh Car, Wheel & Axle Works, Catasauqua, Pa. Besides this description lists are given showing parts of the various sizes and types of mills and illustrations showing installations of Fuller mills in electrically driven and line-shaft driven plants.

¹⁰*Eng. News*, 74, 164-71.

¹¹*Eng. News*, 74, 134-7.

¹²*Canadian Engineer*, 20, 112-3.

Thermal Principles of the Blast Furnace—II

BY J. E. JOHNSON, JR.

In the first article on the thermal principles of the blast furnace I have explained that the heat balance of the blast furnace must be considered not as one, but as two; that there are, in fact, two sets of thermal requirements, one for the preparation of the ore for its descent into the hearth, the other for the production of thoroughly reduced and melted pig iron from the ore so prepared. The present article will show how these amounts may be calculated in detail and will analyze the effect of solution on fuel consumption.

The disposition made of the two kinds of heat had seemed until recently impossible to determine, partly on account of lack of data of actual furnace operations, particularly of accurate average analyses of top gases, since a variation of a few per cent of CO, makes a vast difference in the heat developed. Very recently, however, two excellent sets of such data have been published, one by Mr. Walther Mathesius in his paper before the American Institute of Mining Engineers on "The Use of High Blast Heats in Mesabi Ore Practice," the other by Mr. Sidney Cornell in an article, entitled "The Heat Balance of the Blast Furnace," in METALLURGICAL AND CHEMICAL ENGINEERING for December, 1914.

In the latter the unit of weight used is the furnace charge which yielded in that case 15,260 lb. of iron, in the former the unit used is the ton of iron, but I have reduced them both to the pound basis to which the charted results of Reece's work (Table IV) are directly reducible by simply pointing off two places, it being on the basis of 100 lb. of iron.

In order to reach comparative results we must decide what heat to consider as consumed by the furnace and what as simply developed and then carried out of it. I have adopted the rule that the heat brought in plus that developed less that taken out as sensible heat should be charged to the furnace as consumed.

A curious oversight has crept into the work of both Messrs. Mathesius and Cornell which it has been necessary to correct. They have given the furnace credit for the heat of dissociation of the moisture in the blast, but they have ignored the much larger quantity of heat required for the dissociation of moisture of the charge in the upper portion of the furnace.

Mr. Mathesius takes cognizance of 69 lb. of moisture dissociated per ton of iron from the blast, but the analysis he gives of top gas shows that 240 lb. of water in all are dissociated, there being 27.3 lb. of hydrogen per ton of iron in the top gases. This arises, as pointed out by A. N. Diehl in his paper "Data Pertaining to Gas Cleaning," from the dissociation of the moisture in the ore in the upper part of the furnace. It seems at first sight surprising that this should take place, but it is not when we consider that CO burnt to CO₂ develops 7650 B.t.u. per pound of oxygen, while H₂O to H₂ + O absorbs only 6530 B.t.u. per pound of oxygen.

The heat necessary for this latter reaction has been entirely overlooked by both these writers, with the result that in one set of data the "loss by radiation and unaccounted for" is much larger than it should be. On the other hand, the oxygen arising from this reaction is similarly ignored and the gas analysis given does not account for all the oxygen which must have been present in the top gases.

Calculations based on operating data give us the only means we have of investigating what occurs inside the furnace, and the method of making these is of much importance to show this and simultaneously to obtain the correct figures for the heat developed. We may, therefore, give the calculations based on the excellent data of Mr. Mathesius.

His "Calculations of Operating Results," which contain the data we need, are reproduced here.

CALCULATION OF OPERATING DATA
Analysis of Top Gases

	By Volume	By Weight
CO ₂	14.9	22.3
CO.....	23.5	22.4
H ₂	4.1	0.3
CH ₄	0.2	0.1
N ₂	57.3	54.9

On the basis of above data 1 ton of this gas contains:

$$(2) \quad \frac{22.3 \times 3}{100 \times 11} = 0.0608 \text{ ton of carbon as CO}_2$$

$$\text{and} \quad \frac{22.4 \times 3}{100 \times 7} = 0.0960 \text{ ton of carbon as CO}$$

This carbon originates from:

(a) The carbon charged, minus the amount of carbon transferred to the iron and the amount carried off with the flue dust.

(b) The CO₂ content of the raw materials.

These items are calculated below:

(a) The total amount of coke charged during the period under consideration was

$$(3) \quad 13,006 \text{ tons,}$$

which analyzed:

$$(4) \quad \begin{array}{l} \text{Fixed Carbon} \\ 89.00 \end{array} \quad \begin{array}{l} \text{Ash} \\ 9.40 \end{array} \quad \begin{array}{l} \text{S} \\ 0.59 \end{array} \quad \begin{array}{l} \text{P} \\ 0.008 \end{array} \quad \begin{array}{l} \text{Volatile Matter} \\ 1.35 \text{ per cent} \end{array}$$

(5) The coke as weighed averaged 2.30 per cent moisture, making the weight of dry coke charged:

$$(6) \quad 12,707 \text{ tons.}$$

The amount of fixed carbon charged is thus:

$$(7) \quad \frac{12,707 \times 89.00}{100} = 11,310 \text{ tons.}$$

From this total is to be deducted:

(a1) The carbon carried off with the iron, which is calculated as follows:

(8) The carbon content of the metal was 4.20 per cent and

(9) 17,146 tons of iron were produced, besides

(10) 173 tons of scrap (ladle skullings and pig-machine scrap); therefore

(11) 17,319 tons of iron (17,146 + 173) carried off

$$(12) \quad \frac{4.20 \times 17,319}{100} = 727 \text{ tons of carbon.}$$

Besides the regular burden, the furnace remelted 276 tons of scrap iron, which contained:

$$(13) \quad \frac{276 \times 4.20}{100} = 11 \text{ tons of carbon.}$$

Therefore the net amount of coke carbon carried off with the metallic product is:

$$(14) \quad 727 - 11 = 716 \text{ tons.}$$

(a2) During the month 628 tons of flue dust were taken out of the dust catcher of the furnace. Assuming that the gas carried over an additional 10 per cent of this beyond the dust catcher, the total amount of flue dust produced by the furnace was,

$$(15) \quad 691 \text{ tons.}$$

(16) Since the average carbon content of the flue dust was 8.25 per cent, the amount of carbon carried from the furnace with the dust amounted to

$$(17) \quad \frac{8.25 \times 691}{100} = 57 \text{ tons.}$$

Consequently the net amount of carbon gasified in the furnace is:

$$(18) \quad 11,310 - (716 + 57) = 10,537 \text{ tons.}$$

(b) The only burden constituent containing CO, in larger quantities was limestone, of which 6154 tons were charged. It averaged 43.1 per cent CO, making the amount of CO, charged with the lime-

(20) stone $\frac{43.1 \times 6154}{100} = 2652$ tons which is equivalent

(21) to $\frac{2652 \times 12}{44} = 723$ tons of carbon.

(22) The CO₂ content of the other burden constituents (ore, scale and cinder) averaged about 1 per cent, giving off an additional

(23) $\frac{31,372 \times 1}{100} = 314$ tons CO₂, which is equivalent to

(24) $\frac{314 \times 12}{44} = 86$ tons of carbon.

The total amount of carbon which escaped from the furnace during the month in the shape of CO, and CO₂ is, therefore (see 18, 21 and 24),

(25) $10,537 + 723 + 86 = 11,346$ tons

Each ton of the top gases containing 0.568 ton (see 2) of carbon, the total weight of the dry top gases for the month is:

(26) $\frac{11,346}{0.568} = 72,360$ tons.

The total weight of water, which was charged into the furnace with the ore, coke and limestone, and which subsequently had to be evaporated, leaving the furnace together with the top gases, was, according to the analysis of the burden constituents,

(27) 4665 tons

The nitrogen content of the dry top gases (see 1) originating exclusively from the blast, and the nitrogen content of the air being known, the weight of the dry air which entered into the furnace through the tuyeres can be calculated as follows:

(28) $\frac{72,360 \times 54.9}{77} = 51,592$ tons.

(29) The average moisture of the atmosphere was 5.49 grains per cubic foot at 70 deg. Fahr., making the weight of natural air blown into the tuyeres,

(30) $51,592 + 539 = 52,131$ tons.

The slag volume according to the daily burden (31) calculations averaged 45.5 per cent, making the (32) total amount of slag produced, 7811 tons.

The weight of materials passed through the furnace was as follows:

Material	Total Tons	PER TON OF PRODUCT INCLUDING SCRAP PRODUCED	
		Tons	Pounds
Burden.....	31,645	1.827	4,093
Limestone.....	6,154	0.355	795
Coke.....	13,006	0.751	1,682
Product (including scrap).....	17,319		
Slag.....	7,811	0.451	1,010
Flue dust.....	691	0.040	89
Blast (excluding moisture).....	51,592	2.979	6,673
Moisture in blast.....	539	0.031	69
Top gas (excluding moisture).....	72,360	4.173	9,359
Moisture in gas.....	4,665	0.269	603

According to the average monthly analysis the product consisted of,

(34) C. 4.20 Si. 1.35 Sul. 0.032 Phos. 0.071 Mn. 0.74 Fe. 93.61 per cent

The iron content originated from the following amounts of burden constituents:

Material	Group 1	Group 3	Powable Gmcs	Pit Cinder	Steel Scale	Scrap	Coke	Total
Iron, per cent.....	52.00	49.00	35.00	50.00	66.00	75.00	9.80	
Stage of oxidation.....	Fe ₂ O ₃	Fe ₂ O ₃	Fe ₂ O ₃	Fe ₂ O ₃	Fe ₂ O ₃	Fe ₂ O ₃	Fe	
Pounds per ton of product.....	2,819	755	219	58	207	36	1,682	
(35) Pounds minus 1 per cent loss.....	2,791	748	217	57	205	35	1,665	
Pounds iron per ton of product.....	1,451	386	76	29	136	36	19	2,097

Consequently the iron content of 1 ton of product originated from the following stages of oxidation:

(36) $\frac{\text{Fe}}{39} \quad \frac{\text{Fe}_2\text{O}_3}{165} \quad \frac{\text{Fe}_2\text{O}_3}{1,893 \text{ Pounds}}$

(37) In the same manner it results that 1 ton of product contained 0.74 per cent, or 16.6 lb., of manganese; 16.3 lb. of which have been reduced from MnO, while 0.3 lb. were contained in the scrap charged.

(38) The phosphorus of the product amounts to 1.6 lb. per ton and has been reduced practically all from P₂O₅.

(39) The 1.35 per cent of silicon is equivalent to 30.2 lb. per ton, of which 29.7 lb. were reduced from SiO₂, while 0.5 lb. were contained in the scrap charged.

HEAT BALANCE

(a) Heat Generation

The total weight of carbon contained as CO, in the top gases per ton of product is (see 2 and 33):

(40) $0.0608 \times 9359 = 569.0$ lb.

The weight of carbon per ton of product, equivalent to the amount of CO₂ originating from the limestone and burden and contained in the top gases is (see 21, 24 and 33):

(41) $\frac{809 \times 2240}{17,319} = 104.6$ lb.

Deducting item (41) from (40) gives the weight of coke carbon burned to CO, per ton of product:

(42) $569.0 - 104.6 = 464.4$ lb.

One pound of carbon through combustion to CO, (43) generates 14,543 B.t.u.; the above 464.4 lb. generated, therefore, 6,754,000 B.t.u.

The amount of carbon per ton of product burned (45) to CO is (see 2 and 33): $0.0960 \times 9359 = 898.5$ lb.

One pound of carbon through combustion to CO (46) generating 4446 B.t.u., the above 898.5 lb. generated (47) 3,995,000 B.t.u.

(48) The specific heat of the air blast being 0.248 and (49) the average hot-blast temperature 1248 deg. Fahr., the heat brought into the furnace with the blast

(50) (see 33) per ton of product is: $1243 \times 0.248 \times 6.673 = 2,057,000$ B.t.u.

In the same manner the amount of heat brought into the furnace per ton of product with the moisture (specific heat, 0.49) content of the blast (see 33) is:

(51) $1243 \times 0.49 \times 69 = 42,000$ B.t.u.

(b) Heat Consumption

The following data were used in these calculations:

	B.t.u.
Heat required to reduce Fe ₂ O ₃ to 1 lb. of Fe.....	3,240
Heat required to reduce Fe ₂ O ₃ to 1 lb. of Fe.....	2,970
(52) Heat required to reduce MnO to 1 lb. of Mn.....	2,970
Heat required to reduce SiO ₂ to 1 lb. of Si.....	14,090
Heat required to reduce P ₂ O ₅ to 1 lb. of P.....	10,620

The reduction of 1 ton of product therefore requires the following heat (see 34, 36, 37 and 38):

	B.t.u.
1803 lb. of Fe from Fe ₂ O ₃	6,133,300
165 lb. of Fe from Fe ₂ O ₃	490,000
(53) 16.3 lb. of Mn from MnO.....	48,400
1.3 lb. of P from P ₂ O ₅	17,000
29.1 lb. of Si from SiO ₂	418,500
Total.....	7,107,200

The weight of CO₂ produced by the calcination of carbonates per ton of product is, according to (20, 23 and 33):

(54) $\frac{(2652 + 314) \times 2240}{17,319} = 384$ lb.

practically all of which was contained in the burden in the form of CaCO₃. The heat necessary to de-

(55) compose CaCO_3 into CaO and CO_2 is 1830 B.t.u. per pound of CO_2 . The driving off of above 384 lb. required.

(56) $1830 \times 384 = 702,000$ B.t.u.

The amount of heat carried from the furnace by the slag and iron was not determined. The figures used represent the average results of former tests under similar conditions. According to these the heat carried off by 1 lb. of iron amounted to

(57) 510 B.t.u. and per pound of slag to 900 B.t.u.

The heat carried off per ton of product with the iron amounts to

(58) $510 \times 2240 = 1,142,500$ B.t.u.

and with the slag

(59) $900 \times 1010 = 909,000$ B.t.u.

The dissociation of the moisture as carried into the furnace with the blast requires, per pound of (60) water vapor, 5760 B.t.u.; per ton of product, the heat required for this reaction was (see 33):

(61) $69 \times 5760 = 397,000$ B.t.u.

The dry top gases carried off the following amount of heat per ton of product. (The average top temperature was 325 deg. Fahr., (see 1) for the gas analysis and (33) for the weight of gas.)

	Gas analysis	Weight of gas	Top temperature	Spec. Heat	B.t.u.
	CO_2	0.223	9,359	325	0.2169 147,100
	CO	0.224	9,359	325	0.2426 165,300
	H_2	0.003	9,359	325	3.4090 31,100
(62)	CH_4	0.001	9,359	325	0.5930 1,800
	N_2	0.549	9,359	325	0.2438 407,100
	Total				752,400

The moisture carried out by the top gases entered the furnace at a temperature of 70 deg. Fahr., i.e., with a heat content (see 33) per ton of product of

(63) $(70 - 32) \times 603 = 22,900$ B.t.u.

Since the moisture was heated in the furnace to 212 deg. Fahr., and evaporated, and the resulting steam superheated to 325 deg. Fahr., the heating of the water required

(64) $(212 - 32) \times 603 = 22,900 = 85,600$ B.t.u.

The evaporation required

(65) $964.8 \times 603 = 581,800$ B.t.u.

The superheating required

(66) $0.48 \times (325 - 212) \times 603 = 32,700$ B.t.u.

Therefore the total heat carried off with the top gases per ton of product amounts to

B.t.u.
85,600 (see 64)
581,800 (see 65)
32,700 (see 66)

(67) 700,100

[End of quotation from Mathesius.]

In the following calculation the letter M followed by a number indicates the item bearing that number in the quotation from Mathesius.

Let us start with the fundamental fact that carbon in the gaseous condition whether CO , CO_2 or CH_4 has a volume of 29.85 cu. ft. per pound of carbon.

The total carbon gasified (M25) is 11,346 tons per month and the iron produced (M33) is 17,319 tons per month, or 0.655 lb. C per pound of iron. Therefore the carbon gas per pound of iron is $0.655 \times 29.85 = 19.57$ cu. ft. The carbon gas in percentage of total (M1) is CO_2 , 14.9 per cent, CO 23.5 per cent or 38.4 per cent total. (We can disregard the CH_4 because no account is taken of the volatile matter of the coke, which undoubtedly contains some carbon). 19.57 being the volume of carbon in cubic feet and 38.4 in percentage, the total volume of gas is

19.57 or 50.9 cu. ft. per pound of iron.
0.384

The nitrogen is (M1) 57.3 per cent of this or 29.2 cu. ft.

and as all this comes from the blast of which it constitutes 79.3 per cent, the blast blown must have been 36.8 cu. ft. per pound of iron. This weighs 0.0807 per cubic foot or 2.97 lb. of which 23 per cent by weight is oxygen or 0.683 lb. The oxygen from the charge may be obtained most easily from M53 and 54.

1893 lb. Fe as Fe_2O_3 carries	$\frac{3}{7} \times 1893$	= 810 lb. O.
165 lb. Fe as Fe_2O_4 carries	$\frac{64}{168} \times 165$	= 62.9 lb. O.
16.3 lb. Mn as MnO carries	$\frac{16}{55} \times 16.3$	= 4.8 lb. O.
1.6 lb. P as P_2O_5 carries	$\frac{80}{62} \times 1.6$	= 2.1 lb. O.
29.7 lb. Si as SiO_2 carries	$\frac{32}{28} \times 29.7$	= 33.9 lb. O.
		913.7

From (M54) we find 384 lb. of CO_2 carrying $32 \div 44 \times 384 = 279$ lb. per ton, a total from these sources of $913.7 + 279 = 1192.7$ lb. per ton or 0.5325 lb. per pound of iron. To this must be added the oxygen of the blast 0.683 lb.

The moisture in the blast is dissociated in the hearth and the oxygen is added to that from the other sources and Mr. Mathesius has added this, which is quite proper as far as it goes, but it takes no account of the oxygen which comes from the dissociation of the water in the upper portion of the furnace. The moisture in the blast is 69 lb. per ton of which one-ninth or 7.68 lb. per ton or 0.0034 lb. per pound of iron is hydrogen. The hydrogen in the top gas is, however, 4.1 per cent by volume, while the carbon gas is 38.4 per cent and 19.57 cu. ft. therefore the volume of hydrogen is $4.1 \div 38.4 \times 19.57 = 2.09$ cu. ft. or $2.09 \div 178.23 = 0.0117$ lb. or over three times as much as the moisture of the blast accounts for. This hydrogen can only come from one source, the dissociation of water. The volatile matter of the coke would not account for all of it if the latter were all hydrogen, and such analyses as we have show it to contain little or none of that gas. We must, therefore, take account of the heat absorbed by this reaction and of the oxygen as well as the hydrogen produced by it. This oxygen is obviously $0.0117 \times 8 = 0.0936$ lb. per pound of iron. We have then finally

Oxygen from oxides of charge	0.407
Oxygen from carbonates of charge	0.124
Oxygen from blast	0.683
Oxygen from dissociation of water of charge	0.0940
	1.3081

Oxygen required for oxydizing 0.655 C to CO is 0.655 $\times 1 \frac{1}{3} = 0.874$ lb. $1.308 - 0.874 = 0.434$ lb. O left to oxydize CO to CO_2 which suffices to convert 0.326 lb. of C from CO to CO_2 . Of this $12 \div 44 \times 384 \div 2240 = 0.047$ lb. came from the carbonates as CO_2 and therefore generates no heat. We have therefore $0.326 - 0.047 = 0.279$ lb. C burnt to CO_2 and $0.655 - 0.326 = 0.329$ C burnt to CO .

0.279 C to CO_2 at 14,750 B.t.u.	= 4115
0.32 C to CO at 4375 B.t.u.	= 1437
Brought in by blast and moisture (M)	2,099,000
	2240
	937

6489 B.t.u. per lb. of iron

The heat taken out may be considered in various ways: Cornell considers all the heat taken out whether sensible, latent or potential as having been developed in the furnace and then abstracted, while Mathesius considers the latent heat of evaporating the water of the charge as generated and taken out but not the heat of the unburned gases. On the basis previously adopted of considering only sensible heat as taken out I consider that

the latent heat of the water of the charge represents part of the heat consumed in the furnace which is what we are trying to determine. On the other hand the heat which the gas can develop by further combustion obviously plays no part in the operation and should not be considered. On this basis the rule is that sensible heat taken out should be credited to the furnace, but not latent heat, in order to determine its actual heat consumption.

For convenience the sensible heat of the water vapor is taken as starting at 0 deg. although, of course, it is not actually evaporated until many degrees above that point. The error so produced is immaterial especially as there are necessarily many other errors of greater extent due to uncertainties in our data.

The heat removed on this basis is then found as follows:

The specific heat of furnace gas is ordinarily 0.02 B.t.u. per cubic foot, but in this case, on account of the large percentage of CO_2 , it is about 0.0205. The volume of the top gases is 50.9, the top temperature is 325 deg., and the heat removed per pound of iron by dry gas is $50.9 \times 325 \times 0.0205 = 340$ B.t.u., and the sensible heat of the water vapor is 0.269 lb. per pound of iron $\times 0.46 \times 325 = 40$ B.t.u. a total deduction of 380 B.t.u.

The net heat consumed, therefore, is $6489 - 380 = 6109$, say, 6110 B.t.u. per pound.

To this we should add the heat brought in by the raw materials at, say, 60 deg. Their weight is about $3\frac{1}{2}$ lb. per pound of iron, and their specific heat about 0.18, so the heat introduced in this way is about 35 B.t.u., which is not worth bothering with.

Before proceeding to the discussion of heat utilization we may compare the analysis derived from these figures with those given in the paper. The total carbon gas remains the same as does its percentage of the whole; we have therefore only to divide the total percentage of carbon gas 38.4 per cent in the ratio of 0.326 and 0.329, which gives 19.2 per cent CO_2 and 19.3 per cent CO , as against the initial figures given by Mathesius of 14.9 and 23.5. In the data given by Cornell there are included an estimate and an actual gas analysis, and the CO_2 in the estimate is 12.50 per cent and in the test 13.06 per cent, while my estimate made as above is 14.2 per cent.

We have therefore a considerable discrepancy which we can only account for in one of two ways:

A. The analyses are wrong as to the subdivision of the carbon gas, due to some error of not absorbing the CO_2 completely before proceeding to determine the CO or the like.

B. The hydrogen does not arise from the dissociation of water, but is driven off as such from some portion of the charge, presumably the volatile matter of the coke, which, as before stated, is hardly possible.

Fortunately for the correctness of our estimate of heat development, this uncertainty makes very little difference, strange as this may seem, for if the hydrogen *does* come from the dissociation of water we obtain the corresponding oxygen which must oxidize the carbon to the extent shown with the corresponding development of heat, six-sevenths of which must be applied to the dissociation of the water. On the other hand, if the hydrogen does not come from this source, this heat of dissociation is not required and the oxygen resulting therefrom is not produced to cause the oxidation of the carbon to the corresponding extent.

We find then that the total heat consumption per pound of iron in the three cases is as follows:

Mathesius	Cornell	Reece
6110	6860	6440

Reece's furnace had a considerably smaller output

than the others; in fact, not much over half, while the external dimensions of the shell in his case were not very much smaller than theirs, so that the radiation loss is probably considerably larger per pound of iron. On the other hand, no hydrogen is given in his gas analysis, and if this were correct (which the use of a large proportion of hard ore in his practice renders probable), we have a considerably larger consumption of heat per pound of iron than in Mathesius' case.

The total heat consumed in the Cornell example is greater than in Mathesius' by about 10 per cent, of which a part is to be accounted for by the evaporation of more water and the dissociation of more coke in the shaft, but proper allowance for these items still leaves a discrepancy of 7 or 8 per cent for which we are frankly unable to account except on the basis of errors in the gas analyses which underlie the whole structure and which are easily possible.

It should be noted that this variation of 8 per cent corresponds with one of nearly 40 per cent in fuel consumed in the two cases, and is in a broad way a confirmation of the fact that the heat consumption of furnaces working under very different conditions varies but little, as compared with variations in the fuel consumption.

It is a matter of interest also that the heat consumption in the cases of Reece and Cornell are almost identical if we omit from the latter the heat charged to it for the dissociation of water in the shaft, for which nothing has been charged in the case of Reece for the reasons given.

As to the radiation loss per square foot of furnace surface at different temperatures we know little or nothing; the difficulties in the way of investigation are great, and as we do not know how to improve our construction from the point of view of reducing radiation losses it has not seemed worth while to investigate them, and it has apparently never been done thoroughly.

It is well to point out that in those coke furnaces in which the deficiency of heat is in the hearth—that is, those which need more blast heat than they can obtain—the heat carried off from the hearth and bosh is the only portion whose loss affects the economy of the furnace, because it represents a loss of that portion which is already deficient, whereas any heat lost by radiation from the upper part of the stack represents only the loss of a surplus which would pass off as additional sensible heat in the gas if it were not lost by radiation. One proof of this is the fact that at a number of plants the stock going into the top of the furnace is thoroughly watered down to reduce the flue dust losses, and in some cases the top temperature is frankly controlled by the amount of water used. This water has all to be evaporated in the upper regions of the furnace, which requires a great quantity of heat; but no additional fuel is charged for the purpose, merely the sensible heat in the top gases is reduced.

This condition is so familiar to coke furnace operators that it is hard for them to realize that the opposite condition can ever prevail and that the economy of the charcoal furnace is at least sometimes limited by the heat developed in the shaft. But no one who considers the results of the calculations and experiment given above need ever feel any doubt of this.

Since this was written I have been told by one of the best furnacemen in the country that when the thin-lined furnace was used the cooling of the top was so considerable that the wetting of the ore to keep down flue dust had to be discontinued as it reduced the economy of the furnace. This would seem to be a direct confirmation of my results with charcoal.

We find here a smaller development of hearth heat per

pound of iron in the more economical furnace than in the less economical exactly as we found in a considerably greater degree in the charcoal furnace with wet and with dry ore, for while the dry ore figures were theoretical when made, their results were so remarkably confirmed by practice as to give them complete validity.

We have practically proven in the case of the charcoal furnace that the surplus hearth heat was utilized for relatively low-temperature operations; we shall presently see that the same is almost certainly true in the case of the coke furnace having the excess of hearth heat.

The development of hearth heat in the three cases is obtained from the actual carbon burnt in the hearth multiplied by the heat per pound of coke for the given conditions taken from chart I (page 720 of issue of Oct. 15) and divided by 0.85, the chart being for 0.85 carbon burnt.

The carbon burnt at the tuyeres is simply three-fourths the weight of the oxygen of the blast dry plus that of its contained moisture. We omit that reduced from the Si, Mn, and P, which enter the iron, although these are admittedly reduced only with incandescent carbon in the bosh because the reduction of these does not generate but absorbs heat.

The results are as follows:

Mathesius	Cornell
$\frac{0.512}{0.85} \times 1670 = 1005$	$\frac{0.78}{0.85} \times 1360 = 1250$
Reece	
$\frac{0.57}{0.85} \times 1740 = 1170$	

It will, I think, be conceded that these show a very fair agreement, especially as the slag volume in the data by Mathesius was exceedingly small.

Turning now to the method by which the hearth heat is developed we find that practically 50 per cent more carbon is burnt in the hearth in No. 2 (Cornell) than in No. 1 (Mathesius), but that only 15 per cent more hearth heat is developed by it, and the reasonable assumption is that if No. 2 had been given enough blast heat to increase the hearth heat per pound of coke to that of No. 1, and if the distribution of the charge had been as good, so that the furnace would take advantage of the increase, it would have done as good work as No. 1.

The clause in italics is very important. Cases are numerous enough in which a change in the stock distribution had produced an improvement of 30 per cent in fuel economy. The exact cause of the excess fuel required with bad distribution we do not know, and undoubtedly it varies in different cases, but broadly we may be sure that generally, if not always, it is the result of having to do work in the hearth that should have been done in the shaft, and this must necessitate the development of a surplus of hearth heat, precisely as it did at the charcoal furnace above cited, though in that case this condition was not due to bad filling.

It would seem well worth while in cases where the fuel economy is poor to make a brief analysis of the conditions and determine the amount of hearth heat developed per pound of iron. If it exceeds 1100 B.t.u. when making, say, 1200 lb. slag per ton of iron it is more than likely that the cause will be found in bad filling. In one case in my own experience the removal by better filling, without any other change whatever, of a cold core in the center of the furnace which was not properly heated and reduced by the gases in their ascent, resulted in a decrease of fuel consumption of 20 per cent and an increase in tonnage of 25 per cent.

In order to analyze the consumption of heat in detail in both hearth and shaft, let us assume that under aver-

age conditions the total heat consumed is 6500 B.t.u., and that of this the hearth heat is 1100 B.t.u., then the net heat used in the shaft is 5400 B.t.u. Let us assume that the slag volume is 1120 lb. per ton of iron (half the weight of the iron), and that the limestone charged is 1000 lb. per ton.

Our fundamental "second law" states that the heat above the critical temperature is supplied by the hearth heat, and the obvious corollary follows that the heat required below that temperature is derived from the shaft heat.

With this foundation we can make a subdivision of the gross heat consumption by remembering that only those portions of the charge which are discharged from the shaft down into the bosh take out any heat except that which passes out with the top gases, the volatile elements of the charge are heated to various temperatures in the shaft before they are detached from the non-volatile portion of the charge with which they enter, but the heat so absorbed is all given back to the charge again when they are ascending as a part of the gas column, excepting, of course, that heat which they take out as part of the top gas.

In other words, we have only to consider as credits to the shaft heat amounts which it imparts to the iron and the slag-forming materials in raising them to the critical temperature before delivering them into the bosh, and may disregard the temperature to which their gaseous components are heated when the latter are driven off, because the sensible heat so absorbed is restored to the stock column.

In considering the heat absorbed by the chemical reactions which take place in the shaft we do not need to consider the complex steps through which they pass, but only the final result, and from this point of view they are few and simple; the reduction of the iron ore, the decarbonation of the stone, the decomposition of the water vapor. In addition we have to evaporate the water of the charge and provide for the radiation loss. The shaft must supply the heat to raise to the critical temperature that portion of the coke which is burnt in the hearth, but the shaft recovers all this heat exactly as it does that expended on the volatile components of the charge. The quantities of heat required for these purposes are as follows:

Reduction of the Fe counting 95 per cent Fe in iron	2970
Decarbonation of stone	405
Heating slag forming materials to critical temperature	300
Heating Fe to critical temperature	480
Heating carbon in iron to critical temperature	45
Decomposition 0.1 lb. of H ₂ O from charge	580
Radiation deduced from Cornell and Mathesius	440
Evaporate moisture of charge (1/4 lb.)	280
Total consumption in shaft	5500

But our original figures based in practice were 6400 total and 1100 hearth heat leaving 5300 shaft heat, an amount considerably below the actual requirements as above given.

In order to find the discrepancy let us consider next the utilization of the hearth heat. This is expended in reducing the silicon, in imparting to the iron and slag that portion of their total heat not imparted in the shaft, including the latent heat of fusion, supplying the heat lost by cooling water, radiation, and conduction through the bottom.

We know from Cornell's data the amount of the first loss 170 B.t.u. per pound and I have assumed that the other two would equal this amount which is probably a liberal assumption. I have assumed the silicon to be 1 1/2 per cent and assumed the phosphorus and manganese to be included in this.

No attention is paid to the heat of formation of the slag and the carbide of iron since both those reactions give out heat in small quantities instead of absorbing it.

We have then the following consumption of heat in the hearth:

Reduction of silicon.....	180
Ht. of slag not imparted in shaft (900-600) $\times \frac{1}{2}$	150
Ht. of iron not imparted in shaft (600-480).....	120
Cooling water.....	170
Radiation and conduction through bottom.....	170
	790

No allowance is made for the heat required to decompose the moisture of the blast because that has been taken care of by deducting it from the gross hearth heat.

Adding this figure to 5500 consumed in the shaft we have 6290 which is a very satisfactory check in the totals based in practice, but the distribution is obviously erroneous; we utilize in the shaft more than we produce there and in the hearth less.

Going over the matter carefully we see that the error can lie only in one place. We have assumed that *all* the reduction of iron was done in the shaft and none in the bosh, an assumption which, as we shall later see, is contrary to the observed facts of practice and to the indication of the equilibrium diagram in the last chapter which shows that the final traces of oxygen are only removed from iron oxide in an atmosphere almost free from CO, and at a very high temperature; conditions which are only obtained in the hearth.

We see, therefore, that a considerable percentage of the oxygen of the ore is only removed in the bosh of the furnace and from the consumption of hearth heat and the figures given above we can fairly estimate how much.

The data concerning the hearth heat are the most accurate, for the difference, which is a measure of the correction to be made, depends on the smaller number of items and the best established data.

The 790 B.t.u. accounted for, fall short of the 1100 we know to be consumed by 310 B.t.u., and as the total heat of reduction of the ore is 2970 B.t.u., this constitutes 10 per cent of the total, and we can thus feel reasonably well assured that under such conditions 10 per cent of the oxygen of the ore is removed in the bosh. This 300 B.t.u. subtracted from the original figure of 5500 consumed in the shaft leaves 5200 which is at least as good a check with the 5300 based on practice as we have any right to expect.

The high figure used for the total heat of slag is taken from the best sources obtainable and enables us to account for the facts of practice in regard to the effect of increased slag volume on coke consumption, something it is not possible to do in any other way. The heat imparted to the slag-forming materials up to the critical temperature as given in the table, 600 B.t.u., is obtained by adding the heats of the principal ingredients SiO₂, CaO, Al₂O₃, as taken from Richards' data, and is therefore fairly accurate. Deducting this from the total heat of the slag 900 we obtain a difference of 300 B.t.u., which must be imparted in the hearth by the hearth heat. The quantity of the latter per pound of coke charged (not that burned in the hearth only) is in ordinary practice about 1100 B.t.u. so that nearly 0.3 lb. of coke should be required per pound of slag. The figure commonly used is 0.25 lb., but my observation of practice in different districts where very different quantities of slag per ton of iron are made, leads me to believe that this figure is really about 0.33 lb. This is in good agreement with the figure just deduced on the basis of the critical temperature theory, but can not be squared with any other theory by any possibility. For instance, we see that the combustion of a pound of coke develops altogether under such conditions as we have been discussing about 6500 to 9000 B.t.u., taking even the smaller figure and the same high figure for the total heat of slag 900 B.t.u., 0.14 lb. of coke would be required

per pound of slag, which has no relation whatever with practice.

I call particular attention to this in order to reiterate once more that the basis in which the critical temperature rests is not its reasonableness or agreement with accepted principle in other sciences but that it accounts *quantitatively* for the facts of practice, something which no other theory hitherto proposed can even attempt to do.

The shaft heat has so far been found merely by difference, subtracting from the total the hearth heat which we can calculate quite accurately and whose quantity so calculated checks well with fuel consumption over a wide range of practice, so that a number of furnacemen have come to have much confidence in the results so obtained. But while this is a small fraction of the total heat, to obtain the shaft heat merely by subtracting this small figure from the large one of total heat developed does not seem very convincing. We shall be able to show that the heat consumed in the shaft checks well with that imparted to that region; for this purpose we can use the results of calculations made for showing the results of solution loss which are given later on in this article.

There it will be seen that the heat developed and that brought into the shaft by the hot gases from the hearth supply enough for the requirements of the shaft with sufficient left over to heat the top gases to the temperature at which they are actually discharged in practice for those conditions. The radiation loss may have been assumed a little higher than it should and there are undoubtedly minor errors of constants used and assumptions made but it seems inconceivable that so involved a series of calculations could be made on the basis of authentic data and check so accurately as these do with themselves and with the conditions of practice unless they were substantially correct. Granting this we may surely say that here is a confirmation of the two-fold thermal balance of the furnace which cannot be denied since we see not only that the total heat must equal a certain quantity but that a certain portion cannot possibly be applied except above that temperature: also that the two quantities bear a definite ratio to each other depending upon the conditions of the operation.

The Determination of the Critical Temperature

The early work done for this purpose has been described in the quotation in the early part of this chapter and the critical temperature used in the coke-furnace calculations of this chapter is taken at 2750 deg., the same as the figure used for the early work, not for the sake of consistency but because the considerable amount of work done since seems to indicate that for the conditions of ordinary coke practice that figure is approximately correct.

In a valuable paper by F. E. Bachman before the American Iron & Steel Institute in October, 1914, he gives the results of pyrometric observations made by an expert from the Bureau of Standards which agree better than could have been expected with this figure taken as representing average condition of coke practice.

It is greatly to be hoped that more work will be done along this line, particularly on furnaces working under varying conditions, those making ferro-manganese, ferro-silicon and the like, as well as furnaces in different varieties of coke practice and in charcoal practice, both cold and warm blast.

When this is done we may have more correct figures for the various kinds of practice, but in the mean time 2750 deg. for most coke furnaces seems to be reasonably accurate, though on some rather lean slags the critical temperature is probably as much as 100 deg. lower.

(To be concluded.)

Solution of Smoke, Fume and Dust Problems by Electrical Precipitation*

BY LINN BRADLEY

A statement that the earth's atmosphere contains a practically unlimited supply of materials highly valued by civilized man probably would bring to mind the fact that nitrogen compounds are being extracted from the air and used as fertilizers, or perhaps in connection with the manufacture of explosives. On further reflection one might recall the publicity which has been given to the desirability of recovering certain values, among them ammonium sulphate, from the gases arising from bee-hive coke ovens. This is all very commendable, and more serious attention should be given to such recoveries.

But there are also being discharged into the atmosphere regularly many other materials, the aggregate value of which amounts to many thousands of dollars. An attempt will be made to give an outline of some of these latter problems in such a way as to promote and stimulate interest in them.

This is another field in which the recovery of valuable by-products already reflects credit upon the scientific achievements of the technical men of our country. It is indeed gratifying that in the application of electricity to the solution of such problems this country far outranks any other, a Californian, Dr. Frederick G. Cottrell, having done more to advance both the science and the art than any other individual. This special field of technical industry will be described more fully toward the end of this paper.

The terms "smoke problem," "dust problem," "fume problem" have different meanings to residents of various localities. For instance, "dust problem" has one meaning to the superintendent of a calcium carbide plant, another meaning to the foreman of an iron blast furnace, and still another to a resident of a California desert town. Each of these three little appreciates the other's problem and what it means to him.

"Fume problems" cover a great range, from garbage reduction works to sulphuric acid factories, including silver and gold refineries en route.

Even "smoke problem" is not a specific term always meaning the same thing, because some people are inclined to designate as smoke every gas which they can see. It may suffice for present purposes, however, to consider "smoke problems" as covering the gaseous products from burning carbonaceous material, carrying uncombined carbon particles in suspension in such quantities as to appear black, brown or gray, while "fume problems" cover gases carrying in suspension extremely finely divided particles which have resulted from quick condensation of vapors or gases, as, for example, arsenic trioxide.

"Dust problems" refer to fairly large solid particles suspended in gaseous bodies. The dividing line between "fume" and "dust" is not sharp, and oftentimes collected dust may be a mixture of "fume" particles and "dust" particles.

When the suspended particles are liquid, the term "mist" is generally used. Example: sulphuric acid mist, which contains minute drops of sulphuric acid of any strength. A "mist" is readily produced when saturated steam is suddenly mixed with cool, humid air. This "mist" is the so-called "visible steam." It may contain other water in the gaseous condition. Clouds and fogs are excellent examples of how readily these finely divided particles may be kept suspended in a gaseous body, and at the same time they indicate how futile it is to expect

settling chambers to cause the deposition of all of the small particles carried through them by moving gases. Fogs are produced by two portions of the atmosphere, generally of different temperatures and humidities, commingling and producing a condition of supersaturation, the excess water being "fogged out."

The problems of removing from gases minute particles suspended therein are of great variety, and exist not only in thickly settled industrial communities but also in the sparsely settled States of the Middle West and Far West as well as the South. Requests for a solution of specific problems have been received from north-eastern Canada, Cuba, Mexico, Hawaiian Islands, and far-away Alaska, to say nothing of inquiries from across the seas.

In New York City inquiries have ranged from the collection of diamond chips suspended in a small volume of air up to the collection of cinders and small pieces of coke from gases arising from boilers under which about 3000 tons of coal are burned every twenty-four hours. This volume of gas would fill the entire cubical space occupied by the Equitable Building in about three minutes. It is obvious that the saving of values was the object to be attained by collecting the diamond chips, while in the latter case the cinders and coke particles constituted a serious nuisance in the neighborhood.

Black smoke is so common that its origin needs no further reference than to say that practically every furnace fired with bituminous coal may occasionally offend, while some offend almost continuously due to the method of firing. Railroad locomotive engine-houses (round-houses) are generally causes for complaint from residents of the neighborhood, as there are usually several fires being built or maintained in such manner as to produce large quantities of dense black smoke. Locomotives in rapid motion are not such direct offenders, but the accumulation in the atmosphere of smoke from several hundred such fires soon produces its effect throughout the district served.

Plants in which dry, finely divided materials are handled or grinding operations conducted generally have the problem of confining the dust either for saving values, or for sanitary reasons or for preventing nuisances. Sometimes dusts floating around a plant are very detrimental to rapidly moving machinery. Portland cement plants generally have severe dust problems as they feed finely ground materials into long cylindrical kilns near the point where large volumes of gas are rapidly emerging, and in such a manner that the gases pick up a large quantity of dust. Instances are known where the quantity of dust from a cement plant approaches from 200 to 400 tons per day. This dust is scattered broadcast over the district surrounding the plant.

At iron blast furnaces the finely divided ore is charged in such a way that dusting is facilitated. Reports have been obtained which show that the total amount of iron ore and coke lost in this manner amounts to a very large tonnage yearly, and when it is considered that this dust may be worth from \$3 to \$3.50 per ton the economic loss is better appreciated. The presence of this dust in the blast furnace gas is highly objectionable from other standpoints when the gas is to be used in gas engines for direct conversion into mechanical power, or burned in hot blast stoves or under boilers for conservation of the heat energy. Further, careful calculation has shown that by removing the particles from the gases from one iron blast furnace, without cooling the gases, the value of the conserved heat energy might amount to about \$25,000 per annum. One corporation is said to control at least 100 furnaces. Therefore the saving in this way alone would approach \$2,500,000 an-

*An address made before the First National Exposition of Chemical Industries in New York City.

nually. Now add to this the value of the ore recovered in a usable form, the value of the water used in wet-cleaning methods and the power for pumping it, the value due to better hot-blast stove operations, saving in stove and boiler brickwork, labor, etc., and the economic value of electrical precipitation in this one field of industry can easily be appreciated.

In the large smelteries of the Rocky Mountain States the fume and dust problems are of great importance, not alone because of the strife between the farmers and the smelting companies on account of crop damage but because of the huge economic waste occurring in the operations. It is true that other serious problems are to be met in addition to the collection of the suspended solids before the millenium is reached. Markets must be created or extended; transportation must be arranged; new metallurgical and chemical processes must be developed for separating the components of materials now considered valueless. It is true that rapid progress has been made, but much more remains to be accomplished.

While sulphur is being mined in Louisiana and prepared for the market as a distinct enterprise, and some sulphur and certain sulphur compounds are imported from foreign countries, yet the amount of sulphur being wasted in the smelting of valuable ores is extremely great. It is not unusual for a fair-sized smelter to waste 1000 tons of sulphur daily by discharging it into the atmosphere in the form of a gas sulphur dioxide. And this is being done notwithstanding the fact that sulphur is a very necessary commodity for members of a civilized nation. It is the most difficult to obtain of any of the constituents of sulphuric acid, the basis of chemical industry. Some smelting companies are manufacturing sulphuric acid from their previously waste gases. But there is a limit to the market for the grade of acid they produce.

It is evident that other remedies must be sought. Some serious efforts are being made to reduce the sulphur dioxide to elementary sulphur, as in this form it can be stored, shipped or used more universally than if converted directly into sulphuric acid, which is about three or four times as heavy as the sulphur contained, depending upon the acid's concentration. Where smelting operations are so situated as to prevent shipping acid to available markets, some sulphuric acid is being used to extract values from low grade ores which heretofore did not warrant treatment by older methods. Further progress will undoubtedly be made in the future.

Waste gases at smelteries and refineries also contain metals such as copper, lead, zinc, tin, gold, silver, nickel, and even less commonly known elements such as arsenic, bismuth, selenium, tellurium, palladium, antimony and various chemical compounds containing these. The recovery of these materials often would be warranted for the additional revenue which they would produce even under present circumstances; in others, it is desirable to develop processes for separating the constituents of the collected fume and dust. Investigations at one smeltery showed that metals having a gross value of approximately \$4,000 per day were being discharged into the atmosphere. The expense of collecting, smelting, refining and marketing the valuable ingredients becomes an important item, and thus it behooves metallurgists, chemists and engineers to devise cheaper methods of handling such problems.

The arsenic which can be produced from the waste gases of one Western smeltery would much more than equal the entire United States consumption. In order to allow for placing all of this arsenic on the market further uses must be found for it, otherwise it must

continue to settle upon the ground in the surrounding district or be collected and stored until a use is developed for it.

While discussing markets it may be well to state that price alone does not control the consumption of all product. At various copper refineries on the Atlantic Seaboard some selenium is produced, for example. In the elementary form this may sell for around \$1 per pound, and yet, strange as it may seem, large quantities of it are being thrown into sewers. Contrary to the generally accepted theory, dropping the price to even 10 cents per pound would not find a market which would consume all that is now being wasted. This is because of the peculiar properties attached to this element. It can be used for but few things, and even for these in limited amounts only. When chemical research has discovered additional uses for selenium, and likewise for tellurium, this apparently paradoxical situation may be relieved.

The daily press has contained much in recent months concerning brass and the necessity for high-grade zinc. Extensive investigations in large brass foundries have shown that perhaps 3 per cent of all of the zinc charged into the crucibles is burned off or volatilized and then passes up stacks and is discharged into the atmosphere thus constituting an economic loss aggregating many thousands of dollars per annum. Analyses of material collected from the stack gases showed a zinc content of about 55 to 60 per cent, mostly as zinc oxide. Small amounts of cadmium and other compounds are present, but it seems possible to convert all of this zinc heretofore wasted into high-grade zinc, and this would be a distinct and important gain from an economic standpoint as well as being a source of profit to the owners of the foundries.

In the production of metallic zinc from ore a considerable percentage of zinc is lost as fume. In the industry this is known as "ventilator loss," as all of this white fume passes through large stack-like openings in the building roof just over the bank of retorts. This loss amounts to many thousand dollars per annum.

In refining copper electrolytically, gold, silver and other materials are collected as slimes. When these latter are treated in a furnace for the recovery of the gold and silver considerable of these materials are lost as fume. In one plant near New York City these valuable fumes are being collected by electrical precipitation, and whereas previously the actual losses were several thousand dollars per annum, this has been so reduced that probably it does not now exceed \$300 per annum.

There has been produced at this plant a situation which is interesting in showing how progress comes in stages. Prior to 1912 apparently little if any attention was given to the fumes, since their value was considered to be beyond recall and to be not great in any event. After their recovery had been proven it was considered a very serious matter to allow fumes to escape for more than a few minutes while repairs were being effected, those in charge saying that "it cost \$1 for every minute the fumes were allowed to escape."

At a sulphuric acid plant where a large quantity of acid is concentrated in stills considerable mist is made which is considered a nuisance. Investigation showed that 2 to 3 tons of sulphuric acid were being wasted daily as mist. Here is an excellent example of a combined nuisance and value problem.

At another plant hydrochloric acid fumes had been killing vegetation in an adjacent park, but at present the trees and shrubs are thriving, as the acid fumes are being collected electrically. Fume and dust problems may exist within certain manufacturing operations, such as, for example, in the production of sulphuric acid

by the contact process. Sulphur either free or combined with other elements is burned in roasters, the object being to produce sulphur dioxide. These gases (primarily air plus 6 to 7 per cent sulphur dioxide) generally contain dust and fume and some sulphur trioxide, either free or combined with water, to form sulphuric acid. In order to further oxidize the sulphur dioxide to trioxide the gases are passed over a catalytic substance (ordinarily some form of platinum), and in order that the catalyzer be effective the dust should be removed from the gases and the cleaned gases brought to the catalyzer at a temperature of about 400 deg. C. Furthermore, all arsenic, selenium, antimony, lead, etc., and their compounds must be removed as must nearly all sulphuric acid and water vapor. After the conversion to trioxide has been accomplished the absorption of the product by concentrated acid is quite readily accomplished.

It will be seen, therefore, that the removal of dust and fume must be carried out completely and all other harmful impurities eliminated from the gases. Furthermore, the heat resulting from the combustion of the sulphur in the roaster should not be sacrificed. In this industry, therefore, there are some very important dust and fume problems, the importance of which is not based on values in the dusts nor upon neighborhood nuisances, but rather on the improvement of manufacturing operations. Similar problems exist in other industries.

In ventilating modern public buildings, and even up-to-date factory buildings, it is considered desirable to control the temperature and humidity of the air and remove all dust, bacteria, etc. In one recent installation it has been found that by circulating 50,000 cu. ft. of air per minute in a closed system, removing dust, etc., by electrical precipitation, the saving in steam which would be required for preheating this quantity of air in cold weather much more than pays a good return on the investment. Here it is noted that the primary object was conservation of steam for preheating ventilating air.

One of the most interesting problems in dust removal which has been brought up for consideration in conjunction with an ingenious suggestion pointing to its solution has been in connection with the cleaning and ventilation of the present subways. During the small hours of the night, specially equipped cars were to be operated from one end of the subway to the other, these cars to be provided with air-jets or blowers in front so as to stir up all of the dust on the floor, walls and ceilings. The rear of the car was to be equipped with apparatus for sucking in the air and contained dust. This air was to be subjected to electrical precipitation to remove the dust therefrom and thence discharged through the air-jets at the front. This whole arrangement might well be called a giant, portable vacuum cleaner. Perhaps the problem warrants some such equipment, but it would seem that the tracks should be separated by partitions if the scheme is to work satisfactorily.

Reference is hereby made to the article in the Annual Report of the Secretary of the Smithsonian Institution for 1913 by Dr. Frederick G. Cottrell, entitled "Problems in Smoke, Fume and Dust Abatement," and to an article by the author, entitled "Practical Application of Electrical Precipitation," appearing in the April, 1915, *Proceedings of the A. I. E. E.* These two papers give a more extended account of electrical precipitation and are accompanied by bibliographies, thus enabling anyone interested to obtain further details than are given in this paper.

The statement has already been made that the progress with this particular industry has far exceeded that in any other country. While a member of the faculty

of the University of California, Dr. Cottrell conducted experiments in electrical precipitation, and as these were so successful the work was transferred to near-by smelteries and acid plants. To him must be given full credit for the ingenious and practical method which is used for getting the proper kind of electric field, and for developing rational hypotheses and working bases for solving all kinds of fume, dust and smoke problems by electricity. Dr. Cottrell and his associates, Messrs. Heller, Miller and O'Neill, also of California, generously donated their United States patent rights to the Research Corporation of New York City to be used as an endowment for scientific research. This corporation is controlled by a board of directors, each member of which is giving his time and energy gratis for the benefit of this public movement. This unique organization is described and its objects are fully set forth in a paper, entitled "The Research Corporation; an Experiment in the Public Administration of Patent Rights," which appeared in the records of the Eighth International Congress of Applied Chemistry.

The desire of Mr. R. B. Mellon of Pittsburgh to have the coal smoke nuisance in that vicinity eliminated was one of the motives leading to the establishment of the R. B. Mellon Smoke Fellowship of the University of Pittsburgh under the direction of the late Dr. Robert Kennedy Duncan. Partly as a result of the efforts of those engaged in this investigation, the Messrs. R. B. and Andrew Mellon have established the Mellon Institute of Industrial Research and School of Specific Industries, which has such an excellent mission to fulfill in advancing chemical progress. Mr. Mellon likewise has generously donated to the Research Corporation the results obtained by the Smoke Fellowship.

Consideration of the problems heretofore mentioned should suffice to show that some very interesting and valuable problems exist in which extremely finely divided particles must be separated and removed from gases of various kinds, temperatures and volumes.

It was pointed out that settling chambers do not and cannot adequately answer the purpose, since it is necessary to cause some force to act upon the particles in a readily controlled manner and in such a way that they will move away from the moving gas which serves to keep them suspended and separated. With a proper arrangement of pipes or plates and wires acting respectively as "collecting electrodes" and "discharge electrodes," it is possible to produce a discharge of electricity from the wires to the pipes or plates through gases present between them. For convenience the statement is made that this discharge consists of a number of units of electricity, the term being applied as one would use, say, pints of water.

When this discharge is taking place with sufficient rapidity (i.e., enough units of electricity are flowing through the gas in a unit of time), a beautiful violet light can be seen around the wires. It is true that ordinarily air and other gases are considered as unable to conduct electricity, but in electrical precipitation it is customary to employ very high voltages, such as, say, 75,000 to 100,000. With such voltages as these it is possible to subject a gas to such strains that it is changed from a non-conducting to a conducting medium. Of course, the conductivity is not of the first order of magnitude such as possessed by copper, nickel, etc. The discharge and collecting electrodes are not mechanically connected in any way, but on the contrary are necessarily kept spaced apart and insulated one from the other by high-voltage insulators. A large number of discharge electrodes may be connected by metallic wires or plates, and this is likewise true of the collecting electrodes, it being necessary only to keep these two kinds

of electrodes in proper relation with each other so that the units of electricity are compelled to flow through the air or other gases present between them.

To further explain the phenomenon of electrical precipitation, the suspended particles may be compared to a pith ball in an experiment on static electricity. Each of the particles of dust or fume collects a charge of one or more units of electricity of the same polarity as the discharge electrode. Now since two bodies charged with the same kind of electricity repel each other it would naturally be expected that the particles would be driven away from the wire and drawn toward the pipe or plate (collecting electrode). Enough units of electricity must be discharged from the wire or "discharge electrode" or otherwise produced to charge the particles suspended in the gas as it passes between the electrodes. The more units of electricity upon a particle the greater is the force acting upon it, therefore in practice a strong discharge is maintained. Furthermore, the greater the voltage between the electrodes the stronger is the precipitation force. This makes it desirable to operate at a voltage just below that where a spark or arc would form. Moving gases exert a force upon suspended particles, and the greater the velocity the greater is this force. Hence with a given precipitating apparatus there is an upper limit to the gas velocity at which the suspended particles may be completely removed.

If the polarity of the electrodes is too rapidly reversed, the particles will remain in suspension in the gases. For this reason direct current at high voltages is required, as alternating current at commercial frequencies is not suitable for this work. Large volumes of gases require, as a rule, more power than smaller volumes, but the power consumption is affected by other factors. Static machines do not furnish enough power for commercial installations and are not dependable. However, they give a desirable kind of current and potential for the work. In commercial installations the gases may pass between the electrodes in either a vertical or a horizontal direction. Ordinary black iron pipes about 12 in. or 18 in. in diameter by 15 ft. to 20 ft. long are in general use. These pipes may be connected to a gas chamber at each end in a manner similar to that in a tubular boiler. With this arrangement gases and suspended particles may readily be distributed to the various pipes and through them to another gas chamber having been cleaned during such passages. The cleaned gases may be conducted to other places as desired.

The precipitating apparatus may be designed for either hot or cold gases, and for corrosive or non-corrosive materials, and the product may be collected either wet or dry as conditions may indicate. In treating gases containing particles of acid held in suspension, materials such as lead or stoneware are employed to avoid serious corrosion. Heat-insulating materials may be employed in the apparatus to lessen the loss of heat energy when desired. The gases may be either forced or pulled through by fans or other mechanical means, or if the gases are sufficiently warm natural draft may be employed. When treating gases which are explosive when mixed with air, they are kept under a pressure at least slightly greater than atmosphere until they have been cleaned.

The high-voltage direct current is obtained by transforming low-voltage alternating current into high-voltage alternating current and then converting or rectifying the latter into an intermittent, high-voltage direct current. The amount of power required in commercial installations is relatively low, generally being much smaller than that necessary for any other gas-cleaning system.

A unique advantage of electrical precipitation is its ability to collect even the most minute particles. Another interesting feature is that components of gases of different volatilities may be separated by what has been termed "fractional precipitation." In this method the gases are treated while at a high temperature to remove those materials which are then either solid or liquid, and then after further cooling the gases subjecting them to another treatment in a separate precipitator. Arsenic trioxide of high purity has thus been separated from copper, iron, zinc and other materials. It would be possible to obtain water-free tar in gas plants by this means, and also it might be possible to collect tars of different compositions. This offers a fertile field for investigation at gas plants.

At foundries where old tin cans and other waste metals are melted and poured into sash weights, quite large tonnages of fumes high in tin, lead and zinc are being wasted into the atmosphere. While the gross value of this material may be, say, \$100 daily, the fact that these elements are present in such proportions as to render their separation difficult and very expensive by present methods permits owners of these plants to continue such wastage. Research should disclose some practical and economical way of separating these constituents.

In smelting ores of certain districts for the production of copper, the contained lead is generally eliminated from the copper matte in the converters. This fume has heretofore been wasted, hence the lead in such ores was of no value in so far as it never reached a market. Now that this valuable fume can be recovered and the lead marketed, it should be good practice to charge more such ores into the furnaces instead of studiously avoiding them.

There is a large tonnage of galvanized sheet-iron scrap produced yearly, the zinc on which is generally wasted. In some plants this material is charged into open-hearth steel furnaces and the zinc driven off into the atmosphere as fume. By concentrating this material at certain locations, the recoverable zinc fumes should warrant its collection. A somewhat similar problem exists at some iron-blast furnaces, the ore containing zinc in such quantities as to warrant its recovery from the blast furnace gases. An interesting situation has been found at other blast furnaces which indicates a nice research problem. It was noticed that there were white fumes in the gases from the boilers, these being heated by blast furnace gas, burned after being wet scrubbed. A study of these fume-laden gases showed a quite large tonnage of potash compounds which, if collected and properly treated, would make excellent fertilizer.

Many other instances could be given, but it is believed that these will be ample to convince that air and other gases contain values far beyond and of a greater variety than that which is generally understood. It is hoped that chemists, engineers and others will give more earnest consideration to smoke, fume and dust problems and endeavor to solve them, not only for the sake of better atmospheric conditions and for aesthetic reasons, but also for the values which are to be obtained when their solution is accomplished.

Research Corporation,
New York City.

The Buffalo Foundry & Machine Co., Buffalo, N. Y. has opened an office in New York City in the Whitehall Building, 17 Battery Place.

Metallurgical Clay Goods—The Denver Fire-Clay Co., Denver, Col., has issued Bulletin 100, describing its muffles, crucibles and scorifiers and giving instructions for their care and use.

The Electric Arc in Vapors and Gases at Reduced Pressures *

BY W. A. DARRAH

Present commercial arc lamps operate with few exceptions in an atmosphere rich in oxygen or the oxide compounds of the electrodes. As a result the electrodes are oxidized as rapidly as they are vaporized (if not already composed of oxides as in the case of a magnetite arc), and as the oxides cannot be used again commercially it is necessary to provide means for disposing of them. This cycle of vaporization and oxidation of the electrodes naturally results in their comparative rapid consumption, which necessitates the trimming and cleaning of the arc lamp—one of the large factors of arc-lamp operation.

If it were possible to so design the lamp that the supply of material for the arc did not come from the electrodes but from a surrounding gas or vapor, and if in passing through the arc the vapor were not destroyed, it would then seem to be possible to construct an arc lamp which would not require trimming or materially more attention than an incandescent lamp. A lamp of this type, if it had a white light and good efficiency, might in some cases even occupy the field of the present incandescent lamp.

The research which is described in this paper was carried on to investigate the possibilities of accomplishing this object, and while the results are very promising from a practical standpoint, they are outlined here for their scientific value and should be considered only as a scientific research.

A study of the light efficiency of an arc from a chemical standpoint shows that there is a certain rather limited class of substances which when introduced into the arc in small quantities produce a very great increase in luminosity. Thus the addition of small quantities of cerium or calcium compounds to the carbon arc may increase its efficiency over 300 per cent, and the addition

of titanium (preferably as an oxide) to the magnetite arc similarly produces an increased luminosity without a corresponding increase in energy consumption.

It is a peculiar coincidence that the addition of cerium or calcium to the magnetite arc, or titanium to the carbon arc does not result in approximately the same gain in efficiency. The chemical compounds in which the element appears is apparently of little importance.

A study of these facts led to the conception that perhaps an arc could be maintained in a closed vessel and supplied with these light-producing elements in the vapor form. On investigation this did not seem impractical, especially after a consideration of the liquid "inorganic" compounds shown in Table I, which contains some of the elements which produce highly luminous arcs. The table is discussed at length later on in this

TABLE I—EFFECT OF VAPORS ON ARC

Substance	Boiling Point C°	Melting Point	Effect on Arc	Luminosity	Color
1. Carbon tetra chloride	77°	Liquid at ordinary temperature	Stable	Low	White
2. Stannic chloride	114°	Do.	Stable	Medium	Yellow white
3. Titanium tetra chloride	136°	Do.	Unstable Oscillatory Arc	Very high	White
4. Antimony trichloride	220°	Do.	Stable	High	Yellow white
5. Arsenic chloride	130°	Do.	Stable	Low	Purple
6. Silicon tetra chloride	57°	Do.	Unstable	High	White
7. Boron chloride	18°	Do.	Unstable	High	White
8. Phosphor trichloride	75°	Do.	Stable	Low	Pink and green
9. Aluminum trichloride	183°	Volatile	Do.	Low	Yellow
10. Cerium chloride	1400°	Do.	Do.	Very high	White
11. Calcium chloride	1200°	780°	Do.	High	Strong yellow
12. Calcium iodide	1100°	740°	Do.	High	Yellow
13. Chromic chloride	White heat	Do.	Unstable	Low	Green
14. Titanium chloro bromide	180°	Do.	Very stable	Very high	White
15. Bromine	63°	7°	Very stable	Low (high absorption)	Yellow
16. Iodine	114°	Volatile	Unstable	Low	Blue
17. Chlorine	33°	102°	Stable	High	Yellow

*A paper read before the New York section of the American Electrochemical Society, in joint session with the Illuminating Engineering Society on November 11, 1915. Somewhat abstracted.

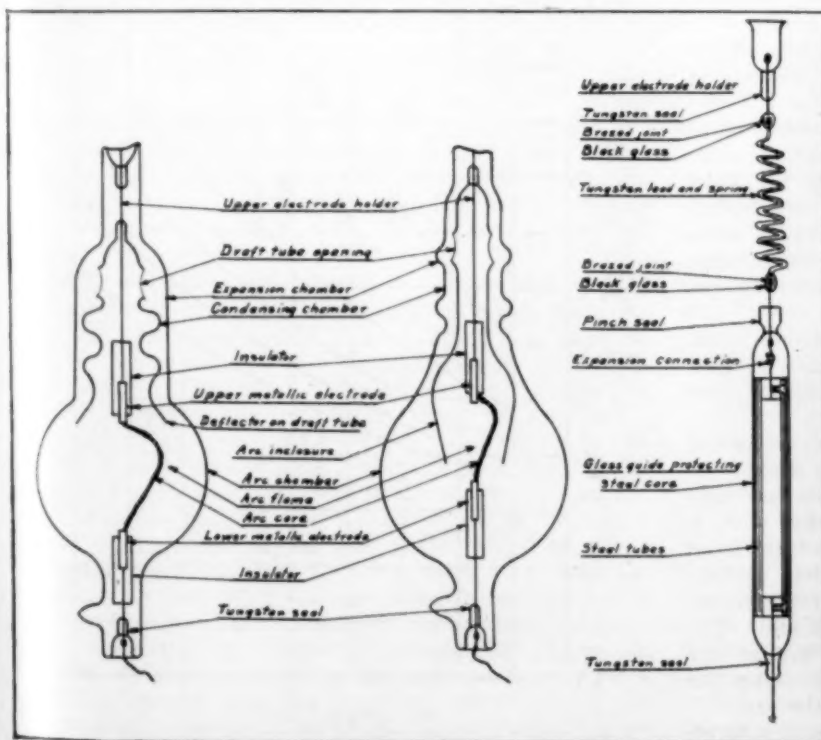


FIG. 1—LATEST CONSTRUCTION OF NEW ARC LAMP TYPE



FIGS. 2 AND 3—VIEWS OF NEW TYPE OF ARC

paper, but it is interesting to note that titanium, tungsten and other elements which increase the luminous efficiency of an arc, may readily be secured in the form of volatile liquids.

Accordingly, a crude apparatus was arranged in which two hard carbon electrodes were placed vertically above each other in a tight glass vessel, and an arc drawn between the electrodes. As might be expected, this arc was non-luminous and very unstable. Accordingly, a small amount of carbon tetra chloride was poured into the vessel with a rather marked result. The arc then became stable and could be drawn out to several times the length which was previously possible, although its luminosity was not much increased. A flaky soot formed by the decomposition of the carbon tetra chloride also filled the glass chamber. A trial was next made with stannic chloride, with a similar increase both in stability and allowable arc length. In this case, also, the luminosity of the arc was much greater. The stannic chloride, however, was decomposed by the air and moisture present, forming a white insoluble powder upon the walls of the chamber, soon shutting out most of the light.

A more elaborate apparatus was therefore constructed in which the graphite electrodes were separated by an electro-magnet acting upon an iron core within a glass chamber.

The air was exhausted from the chamber, a small quantity of stannic chloride vapor at a pressure of from 3 to 4 centimeters was allowed to enter and the arc drawn. Under these conditions a stable arc 3 to 4 in. long could be produced with 6 to 7 amp. direct current, and 220 volts impressed. The arc was intensely white and so steady that it had the appearance of a thick incandescent filament slightly bowed and surrounded by a flame-like less luminous shell of yellow. The vapors from this arc, however, soon collected on the walls, forming a dense brown coat, which proved to be mainly tin carbide with a mixture of stannous chloride and other intermediate compounds.

These preliminary tests having indicated possibilities as well as having shown the difficulties to be overcome, additional smaller lamps were constructed along somewhat improved lines.

With these a detailed study was made of the effects of the various available electrode materials as well as the different vapors and lamp designs.

For convenience a hard glass (manufacturer's number G-702-P) was employed, as this permitted the sealing of tungsten wires and rods directly into the globe by the use of a flux of potassium nitrite and borax. This allowed the free use of heavy seals which were necessary for the tests. Fig. 1 shows the latest designs which have eliminated many of the difficulties encountered.

The shapes shown in Figs. 2 and 3 were found fairly satisfactory, while Fig. 5 shows some later designs which eliminated many of the difficulties eventually encountered.

The type of lamp shown consists essentially of an arc chamber at the center of which the arc is drawn between two tungsten electrodes about 3/16 in. in diameter. The electrodes are partly surrounded with a refractory insulator, designed to prevent the arc from moving far from the ends of the electrodes, as this allows the magnetic blow of the arc to continue to extend it, ultimately tending to rupture the arc. The refractory insulator also assists in maintaining the electrodes at a high temperature, thus increasing the efficiency of the lamp and the stability of the arc.

The upper electrode is fastened by means of a tungsten rod to an iron core, which is surrounded by a glass

shell to protect the iron from corrosion, as the vapors used readily attack nearly all metal except tungsten, platinum, gold and a few other inert materials. A flexible tungsten spiral serves to conduct the current from the upper movable electrode to the upper seal.

It was found after a number of trials that a certain combination of gases to be described later gave a minimum amount of "soot" (decomposition products), which gradually collected upon the globe, thus reducing the amount of light radiated. In order to overcome this difficulty the arc chamber was extended somewhat above the upper electrode, thus forming a condensing chamber in which the soot could collect, allowing the walls to remain clean.

Further study developed that while this would remove a large quantity of the suspended soot a certain amount was still deposited on the glass walls adjacent to the arc. In order to eliminate this deposit, a draft tube was placed around the upper electrode in such a position that the heated gases from the arc would pass upward through the draft tube out into the condensing chamber and then back downward into the arc chamber for use again. The corrugations shown on the draft tube and condensing chamber has the effect of slightly compressing and expanding the gas as it passes from a chamber of small volume to one of large volume. The alternate expansion and compression of the gases, as well as the contact with the walls thus produced, was found to materially aid the condensation of the suspended matter.

The search for a satisfactory electrode material developed that carbon, either in the form of graphite or in the amorphous state, was unsatisfactory because of the disintegration by the action of the vapors and also the deposit of free carbon which it caused on the walls of the arc chamber apparently by direct radiation, as the draft tube and condensing chamber did not affect it as much as an increase of pressure of the gas around the arc.

It was demonstrated that practically all of the materials available (see Table I) formed carbides under the influence of the arc, and these carbides mixed with free

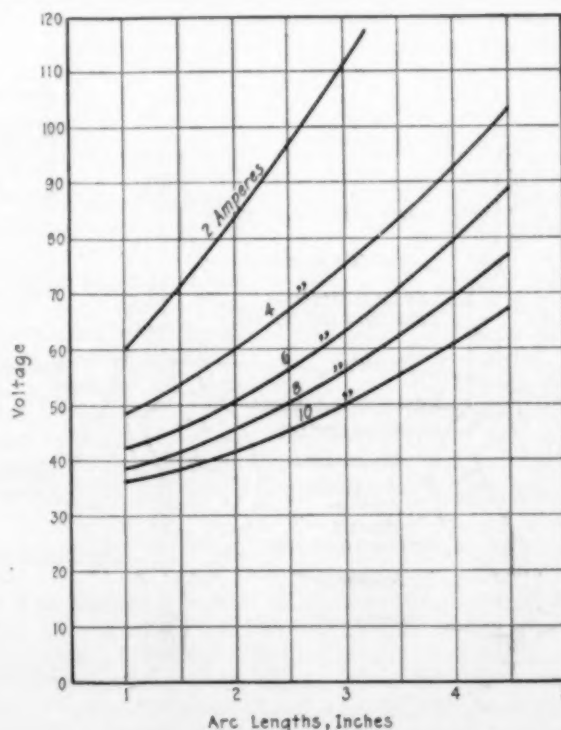


FIG. 4—PERFORMANCE CURVES

carbon formed so heavy a deposit on the walls of the arc chamber that it was decided to abandon carbon.

Chromium, titanium, molybdenum, nickel, iron, and silver were all tried without satisfactory results. Decomposition of the electrodes, decomposition of the surrounding gases and heavy deposits on the globe made the use of these materials impractical.

The substance which was found to be satisfactory in all respects was tungsten, which proved to be practically inert even at a white heat in the presence of the various vapors used. The tungsten terminals, moreover, did not melt or evaporate in the arc except at a very slow rate, which seemed to be comparable to the evaporation of the filament in an incandescent lamp. A rather interesting phenomena resulting from the gradual evaporation and deposition of the tungsten electrode was exhibited in the growth of a thin star-shaped fringe on the edges of the arc surfaces of the electrodes. This fringe was composed of points 2 or 3 millimeters long formed of small needle-like crystals of tungsten, and appears to be caused by the slow precipitation of free tungsten from the surrounding gases under the influence of temperature changes.

Having devised a satisfactory construction it was found that in order to introduce the gases and eliminate air and moisture special methods were required.

As all of the liquids employed are hygroscopic, and many of them readily decomposed by ordinary moist air, it became necessary to exhaust the air from the lamp, heat the lamp and electrodes while at a low pressure in order to remove the air films and then admit the liquids without opening the lamp chamber to the air. This was accomplished by means of a bottle sealed to the lower extremity of the lamp and provided with a ground stop cock connecting it with the lamp, as shown in the photographs. After the proper degree of exhaustion was reached, the desired liquid was carefully poured into the bottles and the stop cock opened slightly when atmospheric pressure forced the liquid into the lamp chamber.

The appearance of the arc in this lamp is quite different from other commercial arc lamps (see Figs. 2 and 3).

It is usually very stable, about $\frac{1}{8}$ in. in diameter, tubular in form and varies from 2 to 5 in. in length with 110 volts direct current applied, the variations being due to differences in pressure, natures of the gases supplied, etc. A high pressure naturally increases the intrinsic brilliancy of the arc, but makes it necessary to operate at a shorter length as the gas currents due to temperature differences and arc reaction are correspondingly more violent. The diameter of the central luminous tube is also reduced with an increase in pressure.

Some of the arcs revolve very slowly, but as a rule the low pressure of the gases minimizes the motion of the arc and the general appearance is that of a thick intensely white incandescent filament slightly bent at the upper end. With some of the vapors, as, for example, the metaloid halogen compounds, antimony, phosphorus and arsenic chlorides, the central tubular portion is surrounded by a ragged flame, which is light pink in the case of arsenic and pale green in the case of phosphorus.

The flame portion acts somewhat as an absorbing screen, thus reducing the total useful radiation. It is a curious fact that this flame may exist for a short but appreciable time after the circuit has been interrupted, thus indicating that it probably does not have a part in the conduction of electricity through the vapor, but appears to be a zone in which the vapors after being dissociated by the heat and electrical effects of the arc recombine. The spectrum of the flame is usually quite different from the arc spectrum, and is characteristic

of the elements involved, while the spectrum of the arc independent of the ends of the electrodes (which, of course, give a hot body spectrum) is usually a band spectrum and is practically continuous except for a few absorption lines.

A curious phenomenon noticed with an 8-amp., 220-volt arc at low pressure in the presence of cerium chloride was a separated arc which gathered into a glowing sphere at one corner of the chamber while the space between the electrodes through which a current of 8 amp. passed was entirely dark.

The complete paper contains typical performance curves which indicate the electrical characteristics of the antimony pentachloride lamp, which is illustrative but does not have as high an efficiency as the titanium tetra chloride or titanium bromo chloride lamp.

Fig. 4, plotted between arc length and voltage, shows the rate at which additional arc lengths require an increased voltage. The curve is also interesting as indicating that a 4-in. arc at 10 amp. requires no more voltage than a 1-in. arc at 2 amp. In other words, a 10-amp. arc is four times as long and therefore emits four times as much light as a 2-amp. arc. In addition its intrinsic brilliancy is about five times that of a 2-amp. arc. Accordingly, the light emitted by a 10-amp. arc *under these conditions* is twenty times that of a 2-amp. arc, and since the wattage of a 10-amp. arc is only about five times that of the 2-amp. arc, the efficiency is approximately four times that of the 2-amp. arc.

Efficiency curves indicate that 20 amp. is approximately the point of maximum efficiency of this particular antimony pentachloride lamp. It should be understood, however, that the current of maximum efficiency differs widely with differences in the nature and pressure of the gas as well as the volume and available radiating surface of the lamp, and very efficient titanium bromo chloride lamps have been constructed with a maximum efficiency of about 5 amp.

Regarding the efficiency secured, it is to be expected that these will vary widely with the condition and nature of the vapors used, and actual results range from 1 watt per mean spherical candle-power to less than one-quarter of the value. This performance may be secured with a white light which closely resembles afternoon daylight (north sky). An analysis of the color of the arc with the spectroscope as well as color-matching tests confirm this fact.

As a matter of interest, Table I shows some of the different compounds tested and some of the general results secured. These compounds may be roughly divided into three classes, as follows:

1. The illuminants, or those materials which give a high luminous efficiency. They usually produce unstable arcs if used alone.
2. The stabilizers, or those compounds which have a low luminous efficiency, but when introduced into an arc allow a very stable arc at relatively low voltages.
3. The "catalyzers," or those compounds which are introduced to assist recombination in the flame zone, thus minimizing deposit.

Table I is interesting as it is the peculiar properties of the compounds, many of which are not very commonly used, that render possible a lamp of the kind described. The problem is to introduce as dense as possible an atmosphere of the various illuminants as titanium tetra chloride into an arc in such a form that it will not be permanently decomposed but may be used again and again. Some of the liquid halogen compounds shown in the table allow this. Some of those compounds (as, for example, those with the metaloids) which have a relatively slight affinity for a halogen are too readily

decomposed to be useful. Others, like stannic chloride, are too readily reduced to a less oxidized state, passing from a tetra chloride to the dichloride. Titanium tetra chloride is an example of this condition, but the addition of a small quantity of free bromide to the $TiCl_4$ (in which it is readily soluble) forms a compound (the bromo chloride) which readily recombines after dissociation.

Fortunately also, since the chloride and bromochloride of titanium dissociate at a higher temperature than tungsten chloride, and therefore recombine at a higher temperature than tungsten chloride, these vapors will not attack metallic tungsten even in the presence of the electric arc.

In order to avoid oxidation effects it was necessary to eliminate all oxides from Table I. More complex compounds such as sulphates, nitrates, could not be considered because of the tendency to dissociate and not recombine. Sulphides, nitrides, phosphides, etc., were eliminated because the chemical stability of these compounds in most cases was too slight to be satisfactory.

Chlorine was the halogen most generally used because of the stronger union of the chloride, and the reduced tendency of the dissociation products to attack the tungsten electrodes.

It would appear that an investigation along the lines here described opens up new possibilities in light production, but it cannot be too strongly emphasized that while the results set forth in this paper offer considerable promise from an applied, practical standpoint, yet they are not presented for consideration as representing a finished device, but merely a research in what seems to be a new and interesting field.

Mr. Paul O. Abbe, New York City, has issued a profusely illustrated catalog (A) describing his crushers, jar mills, pebble mills, mixers, disintegrators and various other classes of machinery.

The American Blower Co. of Detroit, Mich., has issued a bulletin describing Sirocco service in large hotels and office buildings.

Pyrometer Recorder—The Wilson-Maeulen Co., New York, has issued a catalog describing the Tapalog, a multi-record, multi-color pyrometer recorder. A description of this recorder was given in our issue of April, 1915, page 260.

Fuel Oil Engines—Messrs. Samuel L. Moore & Sons Corporation, Elizabeth, N. J., has issued a catalog describing its "Crescent" fuel oil engines for isolated stations. This engine operates on low grades of fuel oil, alcohol, hydrocarbon oil or kerosene.

The Richardson-Phenix Co., of Milwaukee, Wis., has sent us Bulletin No. 50 describing the Phenix force feed lubricator and a new type known as model "T," and Bulletin No. 60 containing information on the new model "M" lubricator.

The Yarnall-Waring Co. of Philadelphia, Pa., reports great activity in recent sales of "Lea" V-notch Recording Liquid Meters. Among the chemical and metallurgical companies using them are the American Smelting & Refining Co., the Atlantic Refining Co. and the Aluminum Co. of America.

Investigations of a Source of Potash—The United States Geological Survey in a report just issued (Bulletin 620-J) has suggested that a possible source of potash may exist in the tailings piled up at the concentrating mills of the big copper mines in the West. The potash, however, is combined in the form of silicate minerals, and its extraction has not yet been accomplished commercially, although for several years it has been the subject of much study.

The Development of Refined Tars for Use in Road Construction and Maintenance *

BY PHILIP P. SHARPLES

The year 1915 marks a decade in the application of refined tar to road treatment in the United States on any extensive scale. Experimental work had been done in desultory way before 1905, but the big influx of automobiles in 1905 made the problem of the maintenance of automobile-traveled surfaces acute and stimulated search after a remedy. For road work tar was first practically used in Nice in 1901 by Dr. Guglielminetti.

The problem of finding the most suitable tars for any given use in road work is not an easy one. The variables besides the tar are so many that it is often difficult to assign the exact causes of any success or failure. The observer in comparing different experiments must know not only the chemical and physical characteristics of the refined tar, but must have knowledge in a technical way of the engineering specifications under which the work was undertaken and an actual knowledge of the weather conditions under which the work was prosecuted with an intimate acquaintance with the climatic and traffic conditions imposed upon the road after completion.

The refining of the tar was the first step forward, and since 1905 practically only refined tars have been used in the United States for road purposes.

At the beginning of the decade, the intelligent study and testing of coal tars and coal-tar pitches for industrial uses had only begun. The rule of thumb was still the rule of the tar distiller and the manufacturer's chemist had presented to him not only the development of a new material but the development of methods of testing, the valuation of raw materials, the methods of factory control and the standardization of the finished product. The success attained has reacted on the whole tar distilling industry in the United States and now every plant of any size is equipped with a good chemical laboratory and one manufacturer even sustains a research laboratory unattached to any plant for investigating the scientific problems constantly arising in the industry.

The tar distiller is at the start confronted by a great diversity in raw materials. The tars may roughly be divided into two great groups, the coal tars and the water-gas tars, which differ essentially in chemical and physical characteristics. The coal tars form the basis for by far the larger percentage of the refined tar used for road purposes. The water-gas tars usually require modification by admixture with coal tars or with other bitumens to make them available for road purposes.

Coal tars are further divided for purposes of classification and manufacture into several groups based on their origin. The tars differ markedly one from another in physical properties. Their characteristics are shown in Table I to which is appended a sample of water-gas tar for the sake of comparison. Although the samples are typical of the groups, other tars in the same groups would show much variation in individual test.

TABLE I—COMPARATIVE ANALYSES OF TARS

Tar From	Specific Gravity at 15.5° C.	Specific Viscosity Engler 100° C.	% Free Carbon	% Oil to Soft Pitch	% Tar Acids on Oil
Horizontal retort gas works, high heat.....	1.293	30.0	37.5
Horizontal retort gas works, low heat.....	1.267	21.8	28.9	13.2	14.0
Inclined retort gas works.....	1.238	14.9	24.3	14.3	21.0
Sunset-Solvay coke oven plant.....	1.187	3.0	10.8	21.8	4.0
Vertical retort gas works.....	1.153	2.1	4.0	28.8	29.0
Water gas plant.....	1.074	1.6	0.3	45.1	0

*Presented at the First National Exposition of Chemical Industries, New York City, September, 1915.

The difference in specific gravity, specific viscosity and free carbon is marked and they vary directly. The amount of oils produced when the tars are distilled to a soft pitch varies inversely. Samples of other tars might not fall so regularly into the groups but the main comparison holds true. The tar acids, on the other hand, are much more irregular, though coke-oven tars usually contain lower percentages than gas-works tars. Oil tars, though usually containing no tar acids, may at times show traces or contain a small percentage of tar acids.

The deduction is easily drawn that with such a variation in raw materials, a uniform refined product is only possible with the command of considerable variety of tars and strict laboratory control. A study of pitches of uniform melting point made from these same tars still further accentuates the importance of selection and control, as shown in Table II.

TABLE II—COMPARATIVE ANALYSES OF PITCHES

From	Specific Gravity at 15.5° C.	Melting Point Cube Method in Water, ° F.	% Free Carbon	% Fixed Carbon	% Ash
Horizontal retort gas works (1).....	1.357	127	46.1	51.5	0.14
Horizontal retort gas works (2).....	1.306	131	33.5	42.0	0.21
Inclined retort gas works.....	1.284	130	30.4	37.9	0.10
Semet-Solvay coke oven plant.....	1.281	133	19.3	28.8	0.48
Vertical retort gas works.....	1.225	145	10.4	19.1	0.23
Water gas plant.....	1.190	139	1.1	23.8	0.06

The specific gravity, the free carbon and the fixed carbon advance in the same order as the raw tars. The ash, however, seems to be independent of the origin and is of little moment in the valuation of the tars. A more detailed examination of the pitches would still further develop these differences and would emphasize even more emphatically the importance of strict laboratory control of both the raw material and the finished product.

Refined tars for road purposes are divided into the following groups according to the use to be made of them:

1. Binders used in the construction of macadam roads.
2. Refined tars for use in road-blanket treatments.
3. Refined tars used in cold-surface treatments.
4. Pitches used as filler in block pavements.

The practice in regard to the specifications covering these materials has gradually crystallized and the following may be taken as typical of the grades:

Road Binder

The specific gravity shall not be less than 1.22 or more than 1.28 at 60 deg. Fahr. (15.5 deg. C.).

The material shall have a melting point, determined by the $\frac{1}{2}$ -in.-cube method in water of not less than 100 deg. Fahr. (37.5 deg. C.); nor more than 110 deg. Fahr. (43 deg. C.).

On distilling 100 grams of the material, no distillate shall come over below 338 deg. Fahr. (170 deg. C.). On continuing the distillation to 600 deg. Fahr. (315 deg. C.) not more than 15 per cent of distillate shall come over and the specific gravity of the entire distillate shall be not less than 1.03 at 60 deg. Fahr. (15.5 deg. C.). The residue from the foregoing distillation shall have a melting point not greater than 165 deg. Fahr. (74 deg. C.).

The free carbon shall be not less than 12 per cent nor more than 25 per cent.

In making the foregoing tests the following methods shall be used:

Specific Gravity

The specific gravity shall be determined by the use of a Hubbard type specific gravity bottle. Approximately 10 grams of the melted material shall be placed in the

weighing bottle and the weight taken after cooling. Freshly boiled distilled water shall be added and the bottle kept in a bath at 60 deg. Fahr. (15.5 deg. C.) until no further contraction takes place. The water shall then be adjusted to the mark and the bottle removed from the bath and weighed. The weight of the material, divided by the weight of the water displayed is the specific gravity.

Distillation

A 250-c.c. Engler flask shall be used and 100 grams of the material taken for distillation. The apparatus shall be set up and the distillation conducted as provided for in the tentative method proposed by the American Society for Testing Materials, Proceedings Vol. XI, 1911, page 241.

The distillate shall be collected in weighed flasks and fractions shall be determined by weight.

Receivers shall be changed when the thermometer records a temperature of 338 deg. Fahr. (170 deg. C.) and when the thermometer records a temperature of 600 deg. Fahr. (315 deg. C.) the flame shall be removed.

Melting Point

A clean-shaped $\frac{1}{2}$ -in. cube of the material shall be formed in the mold, placed on a hook of No. 12 (B. & S. gage) copper wire, and suspended in a 600-c.c. beaker so that the bottom of the cube shall be 1 in. above the bottom of the beaker (a sheet of paper placed on the bottom of the beaker, and conveniently weighted, will prevent the pitch from sticking to the beaker when it drops off). The cube shall remain five minutes in 400 c.c. of freshly distilled water kept at a temperature of 40 deg. Fahr. (4.5 deg. C.) before heat is applied. Heat shall be applied in such manner that the temperature of the water shall be raised 9 deg. Fahr. (5 deg. C.) each minute. The temperature recorded by the thermometer at the instant the material touches the bottom of the beaker shall be the melting point.

In determining the melting point of the residue after distillation, the initial temperature of the water shall be 60 deg. Fahr. (15.5 deg. C.) instead of 40 deg. Fahr. (4.5 deg. C.).

Free Carbon

The free carbon shall be determined by making a hot extraction of 5 to 10 grams of material with C. P. Toluol followed by C. P. Benzol, or C. P. Benzol followed by chloroform as solvents. The extraction shall be made in a Soxhlet, Knorr, Underwriters, or some other suitable extraction apparatus. The distillation of the solvent shall be continued until the washings run through practically colorless.

Refined Tar for Use in Road-Blanket Treatment

The specific gravity shall be not less than 1.20 or more than 1.26 at 60 deg. Fahr. (15.5 deg. C.).

The viscosity, tested by the standard Engler viscosimeter, shall be not less than 125 sec. nor more than 200 sec. for 100 c.c. at 212 deg. Fahr. (100 deg. C.).

On distilling 100 grams of the material, no distillate shall come over below 338 deg. Fahr. (170 deg. C.). On continuing the distillation to 600 deg. Fahr. (315 deg. C.) not more than 25 per cent of distillate shall come over. The specific gravity of the entire distillate shall not be less than 1.03 at 60 deg. Fahr. (15.5 deg. C.). The residue from the foregoing distillation shall have a melting point not greater than 165 deg. Fahr. (74 deg. C.).

The free carbon shall be not less than 12 per cent nor more than 22 per cent.

In making the tests, the methods already described shall be used. In determining the viscosity, the following method shall be employed:

Viscosity

The viscosity shall be determined in a standard Engler viscosimeter at 212 deg. Fahr. (100 deg. C.). Sufficient of the material shall be placed in the viscosimeter to bring the surface of the liquid to a level with the tops of the three levelling points. The time required to discharge 100 c.c. shall be recorded as the viscosity.

Refined Tar for Use in Cold-Surface Treatment

The specific gravity shall not be less than 1.14 nor more than 1.18 at 60 deg. Fahr. (15.5 deg. C.).

The viscosity tested by the standard Engler viscosimeter shall not be more than 250 sec. or less than 100 sec. for 100 c.c. at 104 deg. Fahr. (40 deg. C.).

On distilling 100 grams of the material to 338 deg. Fahr. (170 deg. C.) not more than 7 per cent shall distill over. On continuing the distillation to 600 deg. Fahr. (315 deg. C.) the residue shall be not less than 65 grams. This residue shall be a soft pitch at 60 deg. Fahr. (15.5 deg. C.). If the residue appears hard it shall be tested for melting point and the melting point shall not exceed 140 deg. Fahr. (60 deg. C.) by the $\frac{1}{2}$ -in.-cube method in water. The specific gravity of the entire distillate shall be not less than 1.01 at 60 deg. Fahr. (15.5 deg. C.).

The free carbon shall be not less than 4 per cent nor more than 12 per cent.

Pitch for Use in Stone Block with Fine Sand to Form a Mastic as Adopted by New York City

The coal-tar paving pitch shall be a straight-run residue obtained from the distillation of coal tar and shall comply with the following requirements:

- (a) Melting point shall not be lower than 110 deg. Fahr. nor higher than 125 deg. Fahr.
- (b) Free carbon shall not be less than 20 per cent nor more than 35 per cent.
- (c) Specific gravity at 77 deg. Fahr. shall not be less than 1.22 nor more than 1.30.
- (d) Specific gravity of the distillate to 670 deg. Fahr. shall not be less than 1.06 at 140 deg. Fahr., compared with water at the same temperature.

Pitch for Use in Wood Block as Adopted by New York City, 1914

The coal-tar paving pitch shall be a straight run residue obtained from the distillation of coal tar and shall comply with the following requirements:

- (a) Melting point shall not be lower than 135 deg. Fahr. nor higher than 150 deg. Fahr. The contractor before beginning work on any contract shall obtain from the chief engineer in writing a statement as to the melting point desired for that particular contract and a variation of 5 deg. Fahr. either way will be permitted from this melting point, which, however, must be within the limits, as indicated above.
- (b) Free carbon shall not be less than 22 per cent, nor more than 37 per cent.
- (c) Specific gravity at 77 deg. Fahr. shall not be less than 1.24, nor more than 1.32.
- (d) Specific gravity of the distillation to 670 deg. Fahr. shall not be less than 1.07 at 140 deg. Fahr. compared with water at the same temperature.

The tests governing the different elements are all closely interrelated and the attempt to change one without changing the others should not be made.

This is fully illustrated by Tables III and IV. Table III gives a series of refined tars made from the same raw tar, while Table IV gives three samples made from three tars of widely varying carbon content.

The tests used in these specifications and their significance have been given much study during the decade.

TABLE III

Sample No.	Free Carbon Percentages	Distillation Total to 315° C.	Melting Point, ° F.	Schutte Penetrometer, Sec. at ° F.	Viscosity Engler 100 c.c. at 100° C. Sec.	Float Test at 50° Sec.
5	12.1	21.8	29	94	34
	12.0	19.2	108	40	38
8	14.0	16.4	114	50	58
9	14.4	14.9	86.9	85	60	75
10	17.2	12.7	99.7	90	70	110
11	18.2	10.4	108.7	88	80	170

TABLE IV—RELATION OF VISCOSITY TO CARBON CONTENT

Sample No.	Free Carbon Percentages	Melting Point, ° F.	Schutte Penetrometer at 80° F. Sec.	Engler 100 c.c. at 212° F. Sec.	Float Test at 212° F. Sec.
1	1.4	110	42.2	302	158
2	14.5	109	80.1	298	192
3	39.6	112	144.9	739	337

The specific gravity is of value as limiting to a certain extent the origin of the tar. The method used should be distinctly stated and the temperature at which the comparison with water is to be made should be given. Sixty degrees Fahrenheit is a usual temperature but many specifications call for 25 deg. C.

The viscosity or melting point is the most important test in the specification. It has nothing to do with the origin of the tar since any tar may be brought to a given viscosity or melting point above certain lower limits. It does, however, determine the way in which the material can be used and applied. In determining viscosity, the Engler viscosimeter is generally used for materials which are liquid at ordinary temperatures or which may be made fluid at the temperature of boiling water.

The float test or the Schutte penetrometer is often used for intermediate products while the melting point is generally specified for binders and fillers. Whatever method is used, the details must be specified in full.

The amount of free carbon desirable has been the occasion of much discussion. Practical experience in roofing and waterproofing work has shown indubitably the value of free carbon in pitches used for these purposes. In the case of road tars, it was argued that free carbon was an inert substance and could only be a diluent of the tar. A large number of experiments carried on to determine this point have seemed to indicate that tars very low in free carbon were too much affected by temperature and also more subject to change by weathering than tars which contain a fair amount of free carbon. At the same time an excess of free carbon is not desirable since it tends to make the material difficult to work. With cold surfacing materials, the free carbon must necessarily be kept lower, since its presence in considerable amount reduces the penetration of the material into the road surface.

The distillation test is important in defining the kinds of materials used in manufacture. It should always be coupled with a determination of the specific gravity of the distillate and the melting point of the residue.

The method of distillation should be specified with great detail. Unless this is done, the specification will be of little value since widely varying results may be obtained by different methods.

The development of suitable tars has gone hand in hand with the development of suitable specifications for their use and the development of suitable apparatus for their application to road surfaces.

The development of the whole industry has been a happy illustration of the importance of co-operation. The chemist, the civil engineer, the mechanical engineer and the manufacturer has each played his part. Any one, alone, could not have achieved the whole.

Society of Chemical Industry

Meeting of New York Section

The second meeting of the New York Section of the Society of Chemical Industry was held on Nov. 19 at Rumford Hall, Chemists' Building. Owing to the inclemency of the weather, the attendance was comparatively small, but in interest and liveliness of discussions the meeting was very successful.

Two papers on the program, by Dr. G. R. Fonda, on industrial argon and incandescent lamps, and by Dr. W. H. Walker, on chemical research in the iron and steel industry, had been withdrawn and a paper by Dr. Lewis, not on the program, had been substituted. This was first presented.

A FILING SYSTEM IN THE CHEMISTRY OF AMORPHOUS SUBSTANCES

The paper on this subject by Dr. W. K. Lewis of the Massachusetts Institute of Technology was of considerable interest and quite suggestive. This work is based upon the researches of the author as well as upon the researches of other chemists "on both sides of the water." All solids possessing desirable mechanical properties owe these properties to their amorphous condition. Such solids are plastic and can be molded without strain or fracture.

Thesis 1: All or nearly all amorphous solids are composed of large molecular aggregates.

Thesis 2: The molecular aggregates of most amorphous solids can be conceived as built up of a relatively small unit.

Thesis 3: Usually in an amorphous solid V_n the value of n is not fixed.

Thesis 4: The characteristic properties of all amorphous bodies are plasticity and relative inertness.

Thesis 5: The swelling of amorphous solids in solvents is due to the distension of an insoluble and impermeable residue by the osmotic pressure of a dissolved portion.

Thesis 6: Excessive mechanical manipulation of an amorphous solid ruptures the molecular aggregates.

Thesis 7: The utility of amorphous solids is due largely to their property of plasticity.

Numerous examples were cited to illustrate each one of the above seven theses. The author then took up methods of manipulation of plastics.

Method 1: By mechanical stress. The rigidity of the body is such that it will retain its molded shape without any further manipulation, *e.g.*, clay.

Method 2: Plasticity is secured by rise in temperature, rigidity is restored by cooling, *e.g.*, glass.

Method 3: Plasticity is secured by the action of a solvent, rigidity is restored by the removal of the solvent, *e.g.*, sugar.

Method 4: After manipulation of a plastic we may destroy the plasticity by chemical conversion into a new body less plastic, *e.g.*, ceramics, cement, plaster, etc.

Method 5: The plasticity may be lessened by increasing the size of the molecular aggregate produced by coprecipitation of two amorphous substances existing as colloidal solutions.

Method 6: In the complex V_n (amorphous solid) we can increase the size of the molecular aggregate by increasing n , *e.g.*, bakelite and all other condensation products.

Method 7: In the complex V_n we can increase the size of the molecular aggregate by increasing V , *e.g.*, rubber, linoxylin, oxycellulose, etc. In this case a small change in the unit V produces a great change in the plasticity and resistivity, *e.g.*, addition of sulphur to rubber (vulcanizing).

Those elements which do not combine chemically do not affect its plasticity. We also find that n in V_n decreases after rubber has been in solution, and more S is required to vulcanize rubber that has been in solution than the same rubber before it is dissolved.

Theories are a "filing system for facts," and in the manner sketched above many facts from the leather, starch and other industries may be explained.

In the discussion which followed, Dr. M. T. Bogert commented on the originality and novelty of the scheme proposed by Dr. Lewis.

RESEARCH IN CHEMICAL INDUSTRY

C. F. Burgess, president of the C. F. Burgess Laboratories of Madison, Wis., followed with a splendid paper, bringing out many fine points on a subject that some had thought had become somewhat stale for discussion—research in chemical industry. He started with a delightful little story of how his daughter had recently explained to an inquisitive elderly lady that "father used to be a professor in the University of Wisconsin, but two years ago he quit and got to work now."

Some of the points emphasized by Professor Burgess were as follows:

Twenty years ago the electrical engineering courses were most attractive to young men, but to-day chemistry gets their attention more and more.

Research work pays just as it pays to advertise.

Not all men are successful in research.

The research chemist and the manufacturer must ever be aware of the fact that there are more cases of disappointment than of success in research.

Many factories do not know how to conduct a research laboratory. Very often the laboratory becomes a mere adjunct to the sales department, and the unfortunate result is that the "research chemist" is eventually trained to get only such results in his tests, etc., as will boost the quality of the factory's product. Very often such results are far removed from scientific accuracy.

On the other hand, it is not advisable to separate the research laboratory from the sales department altogether. There ought to be an intimate contact between the two departments so that the research man will keep in touch with what is actually demanded.

The personality of the research man is a most important factor. Furthermore, proper training is essential; experience along mechanical lines is a great asset to the research chemist.

The manufacturer himself has to carry out the initial research work—when he selects the men for his research laboratory.

The average university graduate must give up some of his delusions, yet he must not lose his enthusiasm.

The problem of compensating a research chemist properly for his work is very important. To give a bonus to one man is often an injustice to another man. Further, the research worker must recognize the fact that there is a wide gap between research laboratory and factory when it comes to developing a process and putting it on a commercial basis. From his experience Professor Burgess concludes that when a product originally developed in the research department finally becomes a commercial product, the proportion of profits accruing from it which should be charged to the credit of the research department should usually be not more than 10 per cent. The other 90 per cent are due to members of the sales department and other departments. But a profit-sharing plan is difficult to work out.

The best way of taking care of the cost of equipment and maintenance of a research laboratory in a factory is to have a "development account" which might be entered as a fixed sum in the yearly budget.

ADMINISTRATION OF INDUSTRIAL RESEARCH LABORATORIES

Dr. **Raymond F. Bacon**, director of the Mellon Institute of Industrial Research, Pittsburgh, Pa., then presented in abstract an interesting paper of much originality on "some principles in the administration of industrial laboratories."

"Get your principles right, the rest is a matter of detail."

The proper selection of the director or controlling man is the real problem. He must be far-sighted and broad-minded, and must be of "peculiar normality." The direction of a research laboratory is very much like running a "zoo"—there are all kinds of animals with all kinds of tastes and habits.

In research work the unexpected results are often more important than those carefully planned and correctly predicted.

In a research man a pleasing personality is even more important than training or anything else. He must have "ginger and pep," and must be an "active optimist."

As regards the remuneration of the research man, this may be either a straight salary or it may be simply a bonus, or it may be a combination of salary and bonus. At the Mellon Institute the third scheme has been adopted. By bonus is meant a "small" cash bonus of, say, \$2,000 to \$5,000. Dr. Bacon does not believe in royalties or percentage bonuses.

A research man must be free from restraint and from petty annoyances. He ought not to hold an official position which does not permit him to devote his main energy and time to his research work. In the government institutions promotion to official positions is often a means of recognition, but this is a serious mistake.

Before starting a man upon a certain problem, the Mellon Institute makes it a practice to send the man to the factory which is interested in the problem, and there he remains until he is fully acquainted with the exact requirements of the problem.

The habit of reporting upon research work regularly once a week is very important. Not only do these reports form valuable records, but the man often realizes that he has not worked very hard during the past week. Accordingly, this system of reporting serves as a reminder and acts as a stimulus to do better work next week.

Dr. Bacon described at some length the remarkable equipment of the Mellon Institute. The methods of the Mellon Institute have met with great favor elsewhere, as the Royal Institute of Canada and the Sheffield Institute in England were modeled after the Mellon Institute.

GENERAL DISCUSSION ON RESEARCH

Dr. **A. M. Comey** of the Eastern Laboratory of the E. I. Du Pont Powder Company started the general discussion. He said that at their laboratory they find it absolutely necessary to keep routine work and research work separate and distinct. On the other hand, there is a real advantage in maintaining an amicable and intimate relation between the operating or manufacturing departments and the research laboratory.

In order to counteract any tendency of a feeling of jealousy to spring up among the members of the research laboratory they have found it necessary to make one man responsible for a certain problem or work.

The bonus system at the Du Pont laboratory has worked out very successfully. There are two kinds of bonuses:

Class A Bonus: For work that has brought material benefits.

Class B Bonus: For "zeal" where no direct material benefit accrues.

They train their own men at the laboratory by making them assistants or helpers, and they learn from their superiors. Sometimes it is found after a half a year or so that the man is not well suited for research work, and he is then transferred to one of the other departments, the analytical laboratory, the testing laboratory, etc.

When one of their men has made an improvement or an invention which is about to be carried out on a large scale in the factory, he is instructed to write up a detailed report and submit copies of this report—with his name attached—to various heads of the company. At times it is found advantageous to put the research man in charge of the process in the factory until it is fairly well established.

In a research chemist the combination of personality and common sense is more important than training. The man must be happy in his work and must not be hindered by time clocks and other strict regulations.

The remarks of Dr. Comey were heartily applauded. (The methods adopted by him and his company are usually conceded to be the nearest to the ideal. The laboratory has a splendid reputation.)

Dr. **C. G. Fink** of the Edison Lamp Works, Harrison, N. J., emphasized that an intimate relation between factory and laboratory is absolutely essential if the laboratory is to be of any direct value to the company. The head of the laboratory must realize at all times what the immediate and most urgent demands of the factory are and must be prepared at all times to render every assistance possible.

For example, when the European war deprived this country of potash and other raw material, it was necessary to find substitutes, and find them immediately. Thus soda glass was developed for potash glass for a certain special purpose. The factory usually cannot wait very long, and carefully planned research work covering periods of two, three or more years does not interest the factory. Many research men overlook this vital point in making an industrial laboratory an asset and a help. Give the factory "the next best" thing right off and then either improve upon it in the laboratory or find something better to take its place.

As to compensation, most factory heads do not appreciate the importance of giving the research men due recognition and credit for the work accomplished. "Usually a research man prefers proper recognition to an increase in salary or a bonus. He needs this recognition to keep up his enthusiasm. You cannot compensate a research man as you would compensate a salesman or a broker."

Dr. **A. D. Cushman**, director of the Institute of Industrial Research, Washington, D. C., said that they often find difficulty in getting sufficient financial assistance. "But on the whole we greatly enjoy our work. We look upon it as a most interesting game."

Dr. **J. B. F. Herreshoff** urged that in every laboratory there ought to be some men who have had business experience. Very often a young chemist will run into impractical lines. At their laboratories research has been going for very many years, only formerly instead of speaking of research they used to term it "improvements."

Prof. **M. T. Bogert** of Columbia University said that our industries must not overlook the importance of pure research work carried out at the universities. The investigation of new compounds and of complicated molecular reactions, etc., are problems fully as important as, if not more so, than the problems of the industrial laboratories. In this connection it is a source of gratification to learn of the establishment of the Willard Gibbs Professorship of Research in Pure Chemistry at the Mellon Institute. It is of vital importance to the

best interests of all that the industries assist the universities financially in carrying out research work in pure chemistry.

The meeting then adjourned.

The Physics of the Photographic Process*

BY C. E. KENNETH MEES

The unit of the photographic emulsion is the grain, and in order to study its properties we must study the properties of the individual grains. The grains are held in a sponge of gelatin, which is believed by Quincke to have a two-phase structure, one phase constituting the cell walls, and the other a solution filling the channels and cells themselves. Through these channels in the gelatin the migration of ions can take place at a velocity comparable with their diffusion rate in water, while penetration through the cell walls is a much slower and more difficult process.

Development consists of a reaction between an ion of the reducer and an ion of silver from the dissolved silver bromide, the silver ion being reduced to metallic silver in solution, which must then either precipitate or bring the reaction to an end. Precipitation can be induced only by the presence of a nucleus, and consequently in the absence of a nucleus for any grain, that grain will be undevelopable, the provision of the nucleus being the function of the "latent image." From a consideration of the energy required to produce a latent image in a high-speed photographic material it can be shown that the energy available per grain is of the same order as that required to ionize one molecule of a gas, and it is possible, therefore, that the latent image is composed of grains of silver bromide in each of which, on the average, one molecule has lost an electron by the action of light. The action of light on a photographic emulsion may, therefore, be classed as a photo-electric effect, and, as is the case with other photo-electric effects, the light must have a frequency above a certain critical value, the critical value depending upon the salt of silver employed, being about 400 μ for silver chloride, and 480 μ for silver bromide.

It seems probable that reversal is due to a slow decomposition of the silver bromide and liberation of bromine, which then attacks grains which had been exposed and contained a nucleus, rendering them undevelopable or, at any rate, less easily developable.

The size of the grain has several influences on the properties of photographic materials. In high-speed emulsions, if we assume that all the grains are equally sensitive, then the mass of silver produced for a given exposure will vary as the cube of the diameter of the grain, and the range of the sensitiveness experimentally found in high-speed emulsions can be accounted for by variation in the size of the grain without making any other assumptions.

It seems to be a fact that there is a qualitative difference between high-speed and low-speed emulsions, which must be due to the nature of the grains. If we start with a very fine-grained emulsion and cook it, the size of the grain and the sensitiveness will only increase to a certain limited extent, and in order to get a high-speed emulsion having larger grains and much greater sensitiveness, we must start in an entirely different manner, precipitating the silver of medium grain, which can then be increased in size with a concomitant increase of sensitiveness.

The difference may be that in the low-speed type the silver grains consist of a network of crystals containing

gelatin, while in the high-speed emulsions the gelatin is distinctly separated from the crystals, so that in a low-speed emulsion each of the crystals forming part of the grain must be made developable by the action of the light, while the large grain of the high-speed emulsion requires action only at one point to make it entirely developable, the energy required to form a latent image in a high-speed emulsion being only that necessary for the liberation of one electron per grain, while in the low-speed emulsion several electrons for each grain may be necessary. Inasmuch as the grains of high-speed emulsions are not homogeneous in size, the sensitiveness will depend on the distribution of the grains of different size, and so also will the shape of the plate curve.

With a homogeneous emulsion we should have a simple curve with short over and under-exposure portion, but if the size of the grains varies more about the mean, then a curve having a longer straight line can be obtained, while any special distribution of the grains will be accompanied by a corresponding change in the shape of the plate curve.

The resolving power of an emulsion represents its ability to give sharp images and depends upon the side spreading of the light in the emulsion itself. It can be examined by printing images of the edge of a steel blade on the emulsion by parallel light, the distribution of the image inside of the shadow of the blade being afterwards investigated. It has been shown that there are three factors in the resolving powers: (1) The scattering power of the emulsion for light, (2) the absorption of scattered light by the emulsion, and (3) the photographic factor, which covers the reproduction of the light distribution as density distribution in the developed plate.

The distance between the two lines which will just be separated with a given emulsion will be extremely small for extremely small sizes of grain, because such sizes of grain far below the wave-length of light will scatter very little light; that is, the turbidity will be negligible. As the size of grain rises, however, the turbidity increases very rapidly, and with it the distance necessary to separate two lines, until with a grain diameter a little less than 1 μ the turbidity is high and the opacity still low, so that the necessary separation of the lines is a maximum, and then, as the grain still grows, the opacity grows faster than the turbidity and the lines to be separated can be somewhat closer together. Then, again, with the increasing diameter of the grains the actual space factor comes in, the size of the grains themselves and the distance between them becoming comparable with the distance between the separated lines, and with increasing size of grain this distance must again increase steadily.

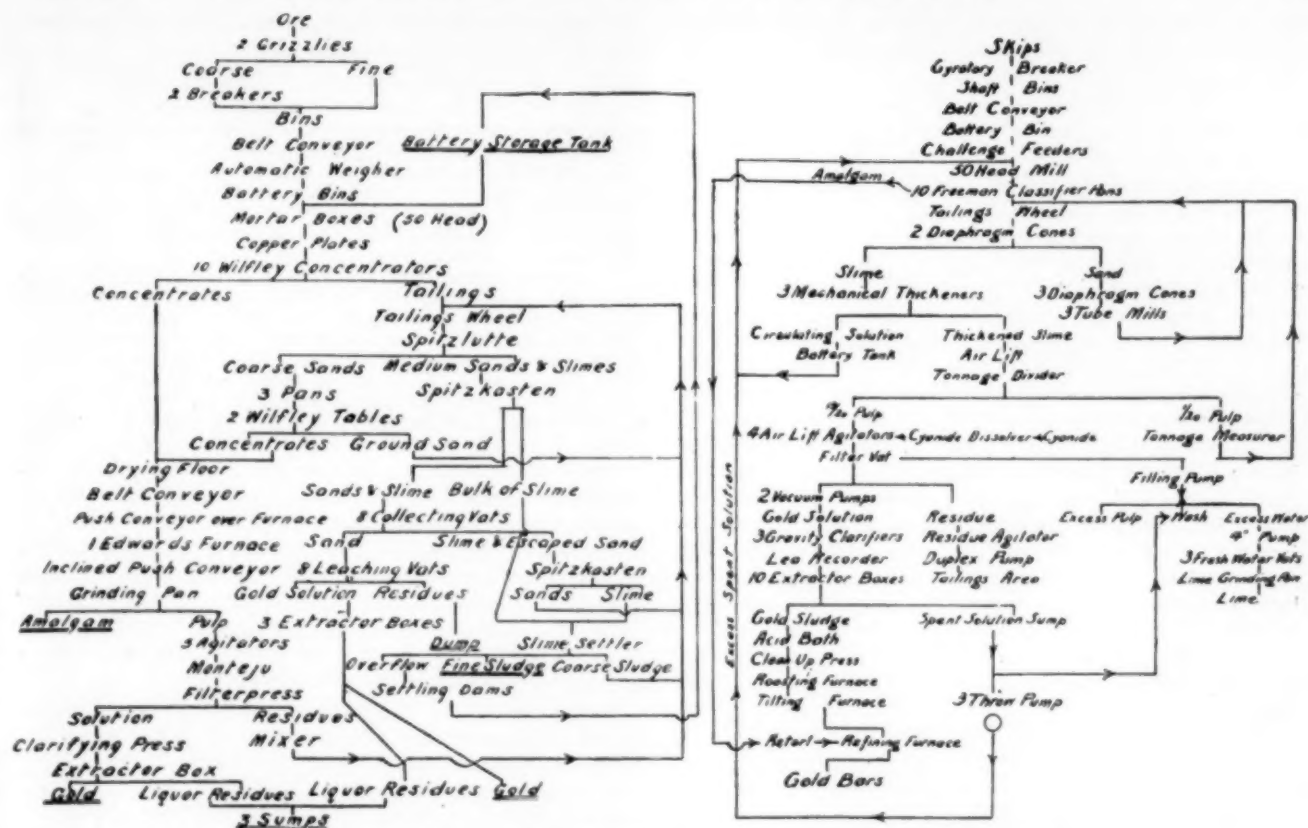
Research Laboratory,
Eastman Kodak Company.

Recent Chemical and Metallurgical Patents

Tungsten and Zirconium

Treatment of Tungsten Ores.—A method of treating scheelite and its concentrates for the removal of phosphorus is disclosed in a patent recently granted to FREDERICK M. BECKET of Niagara Falls, N. Y. A representative sample of scheelite was found to contain 54.60 per cent WO_3 , 3.45 per cent FeO and 0.212 per cent P. The ratio of tungsten to phosphorus was about 204. It is impracticable to eliminate this phosphorus by treating the ore with dilute acids without entailing serious loss of tungsten. The inventor has discovered, however, that if the ore is treated at red heat with concentrated sulphuric acid, the phosphorus is soluble while the tungsten is rendered nearly insoluble. The mixture of ore and acid is allowed to stand for some hours,

*From "Abridged Scientific Publications from the Research Laboratory of the Eastman Kodak Company, 1913-1914. This paper on the physics of the photographic process was originally published in the Journal of the Franklin Institute, 1915.



FIGS. 1 AND 2—FLOW-SHEETS OF 1906 (AT LEFT) AND 1914 (AT RIGHT)

with sand to absorb any slag that may boil over from the crucibles placed thereon. The hearth is covered by a firebrick arch, and an opening at the side is used for placing and removing crucibles. For cupellation a wood-burning muffle furnace is used. The average life of crucibles and muffles is much greater in these wood-burning furnaces than in either coke or charcoal furnaces. Fuel consumption for both furnaces is about 6 tons of wood per month, costing £3 6s. Previous cost of coke and charcoal for the same work was £13 11s., or a saving of £10 5s. per month, to which must be added the saving in crucibles and muffles, amounting to about 35s., or a total saving of £12.

Cyaniding at the Sons of Gwalia, Australia.—The accompanying flow-sheets, Figs. 1 and 2, taken from the monthly *Journal* of the Chamber of Mines of Western Australia, illustrate the simplification of cyanide treatment at the Sons of Gwalia mine. Explanatory data are contributed by Mr. A. WAUCHOPE. The former system consisted in wet crushing, amalgamation in mortar boxes, followed by pan grinding, table concentration, vacuum filtration of slime and leaching of sand. Experiments having demonstrated that all-sliming would be as effective as double treatment, the mill was remodeled on that basis, as shown in Fig. 2.

In the present scheme no amalgamation is done in mortars or on plates, but in 5-ft. Forwood Down grinding pans with Freeman selective pipe discharge. About 20 per cent of the gold is recovered here, with a mercury consumption of 0.23 oz. per ton of ore crushed. Tube-mills have corrugated hard-iron liners, with flint pebbles as the grinding medium. The pebble load for each 18-ft. by 4½-ft. tube is 7 tons, and the consumption is about 1.5 lb. per ton of ore crushed. The slime product contains about 10 per cent of material coarser than 150 mesh. Continuous mechanical thickeners discharge a pulp of 45 per cent moisture which is agitated continuously by combination mechanical and pneumatic

agitators. The filter is of the Cassell type, with forty-four leaves each 9 ft. by 5½ ft. The cotton-duck cloths have a life of about six months, the long service being attributed to a softening of all water taken into the plant. The softening process prevents a precipitation of magnesia when water is mixed with plant solutions. Wet vacuum pumps maintain a vacuum of about 25 in. during filtration and 7 in. during pulp transfer. The elevation is 1400 ft. above sea level. In order to prevent cracking and falling of cakes during changes of pulp and wash solutions, a number of sprays drench the cakes with atomized solution which not only washes the sides of the tank and prevents salting of the following wash, but also reduces loss of dissolved metals. Solution tonnage is measured by means of a Lea V-notch recording meter, which has proved satisfactory over a period of eight months and has permitted close regulation of operations. Solution is precipitated on zinc shavings in boxes having a capacity of 0.8 cu. ft. per ton of solution per day. One and a half tons of solutions is precipitated per ton of ore crushed. The gold-zinc slime is acid-treated, filter-pressed, roasted in a wood-fired reverberatory furnace, and smelted in a tilting, coke-fired furnace. The consumption of chemicals per ton of ore is: KCN, 0.45 lb.; Zn, 0.37; CaO, 3.4; H₂SO₄, 0.2.

Metallurgical Practice on the Rand, South Africa.—A review of the development of gold metallurgy in South Africa was given in a paper before the American Institute of Mining Engineers at San Francisco by F. L. BOSQUI, covering two periods, that preceding the use of cyanidation on a commercial scale in 1890, and the twenty-four years subsequent to that important event. Cyanidation was introduced as a result of trials at the Salisbury battery in 1890 by the Cassel Gold Extraction Co. The first application on a commercial scale was in the same year, when the Gold Recovery Syndicate treated by contract 10,000 tons of tailing on the prop-

erty of the Robinson Gold Mining Co. The Siemens-Halske electrolytic method of gold precipitation was introduced in 1894 at the Worcester mine and was adopted at several other properties, but never won general acceptance owing to the delicacy of the process and the formation of troublesome by-products. It was finally abandoned in 1898 in favor of zinc. With the reversion to the use of zinc, Rand metallurgy may be said to have crystallized into a general scheme of milling and of separate treatment of sand and slime by cyanide. Even the introduction of tube mills in 1904 did not seriously modify the main features, the tube becoming an accessory to the stamp. The vacuum filter was introduced in 1909. The fact that local practice has not adopted numerous processes found feasible elsewhere is not a reflection on the lack of enterprise of Rand metallurgists, but rather of their knowledge of local conditions requiring simplicity in operation, low maintenance and treatment cost, and high efficiency. The ore is low-grade, ranging from $4\frac{1}{2}$ to 9 dw. gold per ton, up to 70 per cent of which is amalgamable. The author gives the following tabulation of variations in local practice as adopted at different mills:

Step in Treatment	Variations
I. Ore sorting and breaking.	(a) Ore washed in trommels or in stationary chutes. (b) Ore sorted on slowly moving belts or on revolving tables. (c) Ore broken in two types of crusher, those in common use being the Blake swinging jaw type and the Gate gyratory.
II. Stamp-milling.	(a) A range in weight of stamps from 1,150 to 2,000 lb. (b) Considerable variation in details of battery construction; also variation in mesh of battery screen dependent chiefly upon tube-mill facilities.
III. Amalgamation.	(a) Use of battery plates, and so-called tube-mill plates for secondary amalgamation after tube milling. (b) No battery plates; all amalgamation done after tube milling. (c) Tube-mill plates, shaking and stationary. (d) Considerable variation in ratio of area of plates to tonnage.
IV. Tube milling.	(a) Tube mills of varying dimensions. (b) Varying ratio of number of tube mills to stamps. (c) Devices for thickening pulp and feeding tube mills. (d) Kinds of "liner" in use.
V. Classification of pulp.	(a) Coarse sand separated out and sent to tube mills. (b) Separation of sand from slime. (c) Classification in spitkasten or hydraulic cones.
VI. Treatment of sand	
1. Collecting.	(a) Collecting tanks superimposed on treatment tanks, or on ground level, and fed by (1) movable hose, (2) Butters and Mein distributors. (b) Collecting tanks filled direct from classifiers or with solution-borne pulp from Caldecott sand-filter tables.
2. Treatment.	(a) Sand dropped from superimposed collectors into treatment tanks. (b) Sand transferred by belt conveyor from collectors on ground level. (c) Sand collected and treated in the same tank from Caldecott sand-filter tables. (d) Sand residue trammed to dump in trucks. (e) Sand sent to dump by aerial bucket conveyor.
VII. Treatment of slime.	(a) By circulation with pumps and decantation. (b) By agitation in Pachuca tanks, followed by filtration with Butters vacuum filter.
VIII. Gold precipitation.	(a) On zinc shavings. (b) On zinc dust in Merrill presses.
IX. Smelting methods.	(a) In Travenor lead furnaces, with cupellation. (b) Direct smelting in pots.

Speaking of the introduction of the Nissen single stamp, the author says that, pound for pound of dropping weight, the Nissen stamp will crush about 30 per cent more rock per day with an increased efficiency of 35 per cent over the ordinary stamp. He sums up the distinct advantages of the single-stamp principles as follows:

1. Owing to the mortar being circular, the screen can be extended around the stamp for the greater part of the circumference of the mortar, so that it is equally distant from the stamp throughout its full length. The screen is therefore always at right angles to the direct splash of the pulp, in the most advantageous position.

2. At each blow of the stamp the pulp is forced radially against the screen, so that all particles sufficiently reduced are discharged. Owing to the mortar box being circular, each time the stamp is raised all the material in the mortar flushes to the center, to be struck by the falling stamp. It follows, therefore, that the uncrushed ore is automatically

returned to the crushing zone, so that the best conditions of feed are maintained.

3. The blow being always received in the vertical axis of the box, the mortar remains rigid on its foundation. This is noticeable in the case of the Nissen stamps at the City Deep, where, after three years of use, the box is apparently as rigid on its base as on the day it was fixed in position.

4. One of the important features of this type of stamp is that it can be cast with a minimum likelihood of shrinkage strains, which are so destructive to ordinary 5-stamp mortars. The foundation bolts give no trouble, not being subject to undue strains.

5. Another advantage of the unit principle is the more continuous operation and flexibility of the entire plan, as each stamp can be put out of commission independently.

6. An interesting feature of this type of stamp is the comparatively even and flat wear of the dies, which results from the return wash of the ore to the center of the mortar with each stroke of the stamp, and from the increased number of rotations of the stamp due to wider cam and tappet faces.

With regard to tube milling, it is now generally accepted that a battery product through a screen of 0.272-in. aperture is about the economic limit of size for tube-mill feed. Beyond this limit the tube encroaches on the domain of the stamp. The proper ratio of tubes to stamps is variable, but in recent practice the tendency is to have a ratio of one 22-ft. 6-in. by 5-ft. 6-in. tube to ten 2000-lb. stamps, or one tube to 200 to 250 tons per day of product of the size given above.

Referring to the development of the principle of peripheral discharge from tube mills, the author gives the following conclusions deduced from tests:

1. That when maintaining the pebble load at the center mark with feeds ranging from 250 to 400 tons per 24 hr., a 25 per cent increase in crushing efficiency could be obtained, but with proportionately increased power consumption.

2. That within certain limits of feed, the weight of the pebble load can be decreased by 25 per cent without affecting the crushing efficiency, with about a 10 per cent decrease in power consumption, the mechanical efficiency showing a corresponding increase.

3. That the wear on the crushing load is increased 300 to 400 per cent.

4. That by decreasing the effective radius of the scoop from the maximum possible (about 29 in. in a 5 ft. 6 in. mill) to the ordinary trunnion discharge, a steady decrease in crushing efficiency is accompanied by a proportionate decrease in power consumption.

Vacuum filtration is now generally conceded to be applicable where it is desired to extend a decantation plant of inadequate capacity or obsolete design; where difficulty in settlement makes it impracticable to use decantation without prohibitive extension of plant, and in all new plants.

Zinc dust precipitation has been adopted at some mines, and offers some distinct advantages over zinc shavings. The following details cover three months' operation at the Brakpan.

	Tons Precipitated (Average per Day)	Gold Content of Dwt.	Gold Content of Dwt.	KCN Content Inflow, per Cent	KCN Content Outflow, per Cent	Alkalinity (CaO) Inflow, per Cent	Alkalinity (CaO) Outflow, per Cent
Sand solution	260	3.952	0.045	0.070	0.036	0.023	0.027
Medium sand solution	799	1.548	0.056	0.017	0.009	0.0146	0.015
Slime solution	1791	1.271	0.048	0.010	0.007	0.0149	0.015

The consumption of zinc in the Merrill presses is approximately 1/10 lb. less than in boxes. At Johannesburg zinc shavings cost 4.2d. per pound and zinc dust 3.93d. (\$0.084 and \$0.0786 respectively). The lower cost of dust is offset by the greater strength of cyanide solutions required for good precipitation. The general opinion is that the zinc dust method is effecting economies in zinc consumption and labor and in affording a complete clean-up of gold.

Percentage extraction has increased from 60 per cent in 1890 to a maximum of 96 per cent in 1914. The cost of reduction of Rand ores ranges from \$0.68 to \$1.20 per ton milled. Figures for the Modderfontein B are given below:

COST OF TREATMENT AT MODDERFONTEIN B.—NOVEMBER, 1914. 42,000 TONS MILLED

	Per Ton Milled, Cents
Ore sorting and breaking.....	3.3
Transport from crusher station to mill bins.....	0.7
Stamp milling.....	17.5
Tube milling.....	12.5
Amalgamation.....	3.6
Cyaniding.....	29.0
Total.....	68.6

Amalgamation Tests.—A discussion of the various proposed methods for estimating the gold content of ores by amalgamation, together with a proposed standard method by the author, W. J. SHARWOOD, formed the subject of a paper at the meeting of the American Institute of Mining Engineers in San Francisco. Twenty-six published methods are reviewed and the following standard amalgamation test suggested:

Equipment Required for Systematic Amalgamation Tests

Agitator. A shaking box, stoutly built, with twenty-four or more compartments, each 3 in. square and 6 in. deep, driven from a shaft running about 250 r.p.m., giving a horizontal travel of about 2 in. The entire bottom, or each section, should be covered with heavy belting or linoleum.

Twenty-four "citrate magnesia" bottles, with spring clamps and rubber washers. These have a capacity about 350 to 370 cc.

Twenty-four Griffin beakers, capacity about 250 cc.

Two enameled-iron pans, rectangular, with straight sides, to hold twelve beakers each.

Filtering rack with twelve 2½-in. funnels and twelve extra beakers.

Hot plate.

Glass-stoppered burette for pure mercury, standing over an enameled pan.

Robervahl scale, turning with ½ g.

Cylinder, graduated to 100, 150, and 200 cc.

Weighing scoop and steep-sided copper funnel or hopper for charging ore into bottles.

Six-inch funnel for water.

A few porcelain dishes about 3 in. diameter, and some enameled pans about 8 in. diameter and about 2 in. deep.

Filter paper, 12.5 cm., S. and S. No. 595 or similar quality.

Scorifiers, 2 in. or less in diameter; these may be used repeatedly and are best glazed before first using.

Washbottle.

Test lead.

Nitric acid, diluted with about double its volume of water.

Mercury, pure or previously assayed.

Silver foil, cut in pieces of about ½ g. each.

Routine Method for Gold Only

One hundred grams of crushed ore or tailing are weighed out, charged into a bottle by means of the metal funnel, 150 cc. of water added, then 1.5 to 2 cc. of pure mercury from a burette. The stopper is then clamped, the bottle folded in a thick piece of cloth or flannel and put into the moving "shaker" for 2 hr. It is then removed, opened, covered with the thumb, shaken and inverted over a 3-in. porcelain dish, and as much as possible of the clean mercury allowed to run out. It is then shaken again for a moment, and if necessary a little more water is added, and more mercury allowed to run out into a second dish, and so on, as long as any mercury comes out. If not floured the mercury is nearly all removed in two operations. The clean mercury thus obtained is transferred to a beaker. The bottle is again shaken well and inverted so as to let a little sand run out into the dish; this is then panned into an enamel dish. Usually only a few globules are obtained; if much is found the whole charge must be panned. If "floured" a small globule of liquid sodium amalgam should be added at this point.

To the mercury, after collecting in a 250-cc. beaker, is added ½ g. of pure silver foil. If this has been carefully prepared and cleaned by treating with cyanide solution or weak nitric acid, or by slightly amalgamating the surface, it may be used to pick up the small globules of mercury obtained in panning. About 150 cc. of nitric acid (sp. gr. 1.14 or 1.15) previously warmed to about 70 deg. C. is now poured into the beaker; all the beakers are set together in a

pan, which is placed on the hot plate and left until all the mercury has dissolved. If not too hot it is unnecessary to use covers. As soon as the mercury disappears the liquid is filtered, the residue is rinsed on to the paper, washed once or twice with very dilute nitric acid (not over 5 per cent) and once with water. The paper is then sprinkled with test lead, folded, and placed in a small glazed scorifier, and covered with lead enough to make up the total to 15 or 20 g. Enough pure silver is added to insure ready parting, and 2 or 3 g. of borax glass. The scorifiers are then charged into a muffle, the paper carefully burned, the lead scorified a few minutes, poured, and cupelled. The cupel buttons are parted in porcelain cups, and the gold annealed and weighed.

A correction is subtracted for gold contained in the silver and mercury.

Using a 100-g. sample, each milligram of gold represents 0.29166 oz., or \$6.03 amalgamable gold per ton.

The author suggests that if assay tons are used, each milligram indicates exactly 0.25 troy ounce, or 120.5 g./when each milligram indicates exactly \$5 per ton. The ore should be ground quite fine, 100-mesh being found suitable.

A Rule Governing Cupellation Losses.—A paper by W. J. SHARWOOD giving an empiric rule by which cupellation losses of silver or gold can be calculated and a correction applied. The rule is thus stated:

When a given amount of silver (or of gold) is cupelled with a given amount of lead, under a fixed set of conditions as to temperature, etc., the apparent loss of weight sustained by the precious metals is directly proportional to the surface of the button of fine metal remaining.

If the above is true the following also must be true:

The loss of weight varies as the 2/3 power of the

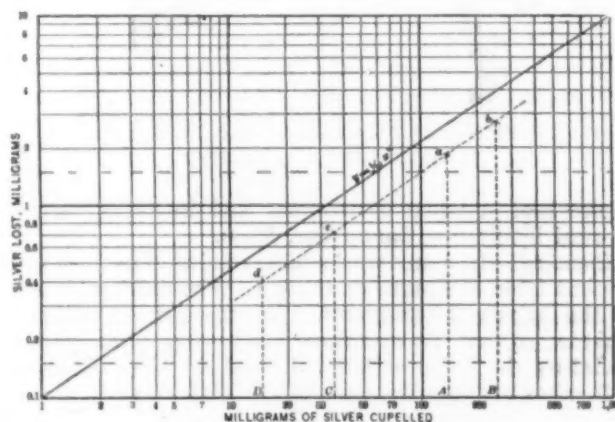


FIG. 3—CUPELLATION DIAGRAM

weight, or as the square of the diameter of the button.

The percentage loss varies inversely as the diameter of the button, or inversely as the cube root of the weight.

The author states that the examination of a large number of experimental results proves the last statement to be a close approximation to the truth, and therefore proof of the others.

"The readiest proof of the rule, which avoids all calculation, is by the aid of logarithmic paper—that is, paper on which the co-ordinates are plotted on a scale like that of a slide rule. In place of using the ruled paper the distances may be actually laid off by means of a slide-rule scale.

"As to the possible application of this rule: An approximate correction may be obtained for an ore or the like, if one proof, as near the expected weight as possible, is cupelled in each row, and its exact loss noted. The corresponding point is plotted on the diagram (a in Fig. 3, A representing the final weight), and a straightedge is placed on the diagram, or a line drawn through a, parallel to the guide line, $y = x^{2/3}$. Noting the points on the line or straightedge corresponding to the weights B, C, D, of other buttons weighed, the correction for each is read at once from the scale. In the example plotted in Fig. 3, a button of 140 mg. weight has lost 1.8 mg.; from this it appears that other buttons of 250, 35, and 15 mg. will respectively have lost about 2.85, 0.72, and 0.40 mg."

Improved Apparatus for Photomicrography of Metals

The chief characteristics of an efficient photomicrographic apparatus are rigidity, firmness, compactness, brilliant and uniform illumination, accuracy of detail in the photograph of the structure under examination, quickness and ease of manipulation.

These essentials are embodied in the improved photomicrographic apparatus, illustrated in Fig. 1 and de-



FIG. 1—STANDARD PHOTOMICROGRAPHIC APPARATUS; ABOUT 1/20 ACTUAL SIZE

signed and supplied by Sauveur and Boylston, Metallurgical Engineers, Cambridge, Mass.

Rigidity.—In many works laboratories vibrations are present from one cause or another. The different parts of the apparatus described here are rigidly clamped together so that the apparatus vibrates as a whole thus obviating the fuzzy or blurred detail so often obtained with other instruments especially at high magnifications.

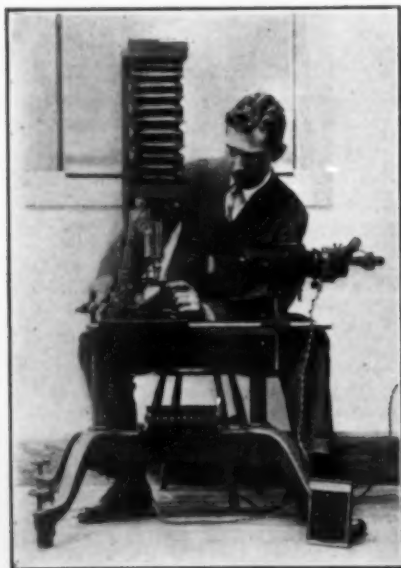


FIG. 2—PHOTOMICROGRAPHIC APPARATUS WITH OPERATOR SEATED

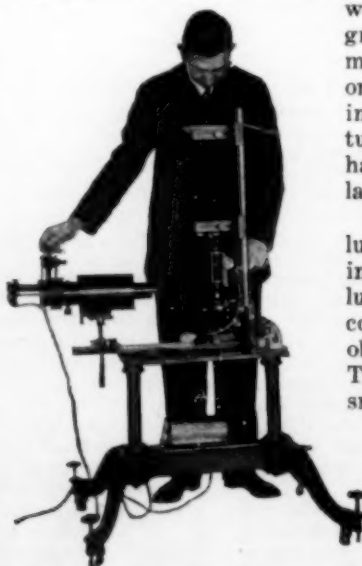


FIG. 3—PHOTOMICROGRAPHIC APPARATUS WITH OPERATOR IN POSITION FOR FOCUSING IMAGE AT CAMERA

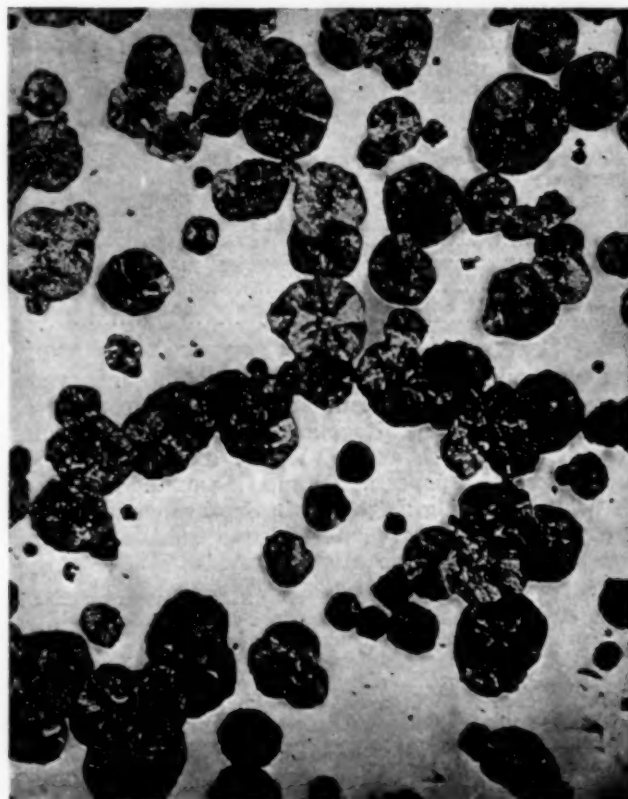


FIG. 4—MICROSTRUCTURE OF QUENCHED HIGH-CARBON STEEL (ROSETTE OF TROOSTITE ON LIGHT BACKGROUND OF MARTENSITE MAGNIFIED 150X) TAKEN WITH APPARATUS ILLUSTRATED IN FIG. 1

Firmness.—The heavy construction of all parts of this apparatus makes for firmness, which in turn means simplicity in adjustment of parts such as the microscope, camera, and arc lamp adjustments.

Compactness.—One of the chief faults of older types of instruments of this sort and of some recent types is a tendency to spread out the apparatus over a large floor space. In this instrument the carbon adjustments and those of the condenser are easily accessible to the operator who is seated in front of the microscope for visual work (see Fig. 2). When taking a photograph the operator simply rises and without moving from the spot can focus the image on the ground glass of the camera in a standing position as shown in Fig. 3. This feature will be readily appreciated by those who have worked with the less convenient or larger apparatus.

Illumination.—Brilliant and uniform illumination of the specimen is brought about in the usual way by the use of a vertical illuminator of the plain glass reflector type in combination with long-mounted achromatic objectives of Bausch & Lomb manufacture. The source of illumination may be either a small arc lamp consuming 4.5 amp. or a 250-watt nitrogen-filled Mazda lamp of special design in light-tight lamp box with the necessary lateral and vertical adjustments for the lamp as a whole. Either lamp may be used on an ordinary house circuit.

Accuracy of Results.—Photomicrographs shown in Figs. 4 and 5 magnified 150 and 400 diameters respectively will give some indication of the accuracy



FIG. 5—MICROSTRUCTURE OF PEARLITE (MAGNIFIED 400X) TAKEN WITH APPARATUS ILLUSTRATED IN FIG. 1

of detail in the structures photographed with this apparatus. These photomicrographs were obtained with the aid of a green monochromatic ray filter regularly supplied with the apparatus together with orthochromatic plates. The use of this screen and plate in combination with achromatic objectives of great flatness of field such as are furnished with this outfit renders the use of expensive and fragile apochromatic objectives of doubtful value.

Speed of Working.—With the green ray filter, orthochromatic plates, and achromatic objectives used in connection with the 4.5 amp. arc lamp on an alternating-current circuit the photographic exposures necessary range from seven to twelve seconds at 100 diameters and from thirty to sixty seconds at 1000 diameters depending upon the contrast in the structure of the specimen. With direct current these exposures are reduced about 30 per cent while with the Mazda lamp the figures given above should be multiplied by $1\frac{1}{2}$ or 2. As many as eighty photomicrographs at 100X have been taken, developed, and fixed in an eight-hour day by one operator using one of these instruments.

The Microscope.—This part of the apparatus is the same in general design as that used by Professor Sauveur as far back as 1892, but with improvements in the mechanical and optical details from time to time which have brought it up to its present standard. The microscope as now furnished includes Sauveur vertically-movable stage for coarse adjustment with the latest Bausch & Lomb lever fine adjustment and mechanical stage. An improved form of magnetic specimen holder* has bevelled edges, which allow the close approach of a high power objective to a specimen without the necessity of a templet or other auxiliary device. Specimens measuring from $\frac{1}{8}$ in. or less in diameter up to 1 in. in diameter can be easily examined by the use of this magnetic holder, which is illustrated in Fig. 6, or by means of the bridge type of specimen holder designed by Sauveur in 1892 and illustrated also in Fig. 6. The microscope has a body tube of $1\frac{1}{2}$ in. diameter, which increases its usefulness when very low power photo-

micrographs are being taken and the new construction of the stand gives a very long working distance between the objective and the stage. For the examination of very large specimens, such as rail sections, etc., an auxiliary tube is provided, the specimen being placed across the horseshoe base of the microscope.

The Camera.—This is generally furnished to take 5 x 7 plates or smaller, and a 4 x 5 camera can be sup-

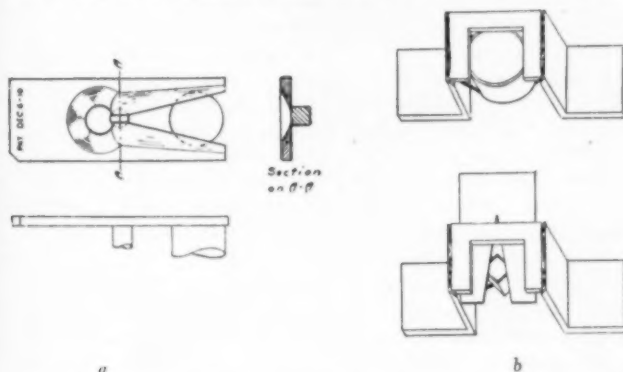


FIG. 6—*a*, MAGNETIC SPECIMEN HOLDER WITH LARGE AND SMALL SPECIMENS; *b*, NON-MAGNETIC SPECIMEN HOLDER WITH LARGE AND SMALL SPECIMENS

plied if desired. The camera is fitted with automatic shutter with Iris diaphragm.

Range of Usefulness.—This apparatus may be used not only for the photomicrography of all kinds of metals and alloys, but also in the case of other opaque objects such as ores, rocks, concrete, slags, wood, leather, paints and varnishes, rubber, etc. The lamp may also be tilted at an angle for use with oblique illumination for opaque specimens, including core and molding sands, and with transmitted light for the examination and photomicrography of transparent sections.

The apparatus is wholly American-made and of moderate price.

Thermit Welding of Camshafts

Experience with thermit welding of camshafts at the Belmont mill, Tonapah, is given by the superintendent, A. H. Jones, in his paper read at the meeting of the A. I. M. E. at San Francisco. Two shafts were welded at the following cost per shaft:

Labor	\$16.00
55 lb. thermit	21.60
Iron punchings	0.38
Wax	1.75
Gasoline (preheat)	2.80
Total	\$42.53

The first shaft repaired was put into service July 1, 1914, and broke, in the weld, Aug. 8, 1914. The second was put into service Aug. 8, 1914, and broke, in the weld, on Sept. 13, 1914. These two shafts were welded at the break, necessitating their being turned down to original size, which is against the advice of the "thermit" manufacturers. On April 6, 1915, another shaft was welded from two long, discarded ends, sawed so that the weld would come in the middle box, and a boss $\frac{1}{2}$ in. high and 4 in. wide left, which was taken care of in babbitting the center bearing. This shaft is still in service and may show a life really worth while. Figuring 8.88 tons per stamp day, the 30-day service from the first two shafts welded would cost about 2c. per ton of ore stamped, which, in case of breakage with no spare at hand, would be better than having ten stamps out of commission for any length of time, and therefore thermit supplies are kept on hand for this and other emergencies.

*Patented Dec. 6, 1910.

New Flotation Testing Machine

The widespread interest that has been aroused in the flotation process of concentration has created a demand for simple and efficient means of making tests on small lots of ore with different oils and oil mixtures. To meet this demand the Denver Fire Clay Company of Denver, Col., has designed the flotation testing unit illustrated in Fig. 1. Neat in appearance, compact and well constructed, it presents a marked contrast to the home-made testing apparatus with which many investigators have had to content themselves.

The Case machine consists of a single aluminium casting comprising the agitation cell and spitzkasten, mounted on a heavy board which also carries a bracket for the support of the propeller shaft and a shelf for the motor. Connection between the bottom of the spitzkasten and the agitation cell, for circulation of pulp, is made of rubber tubing which is readily removed for cleaning the apparatus. The propeller blade is made of aluminium, while the steel shaft is coated with lead in order to resist the attack of the acid that may be used in the process. The motor is a $\frac{1}{8}$ -hp. Westinghouse machine operated from a 110-volt circuit at 1800 r.p.m. Variable speed of the flotation propeller is provided for by a three-speed pulley which will give approximately 1800, 1600 and 1400 r.p.m.

The entire apparatus stands about 4 ft. high and can be attached to the wall or other support convenient to a water supply. Accessory apparatus has been prepared for use with the machine, in the form of a metal hoe or scraper for removing the froth as it forms, and ground-glass-stoppered dropping bottles for measuring the quantity of oil used. The machine will prove a

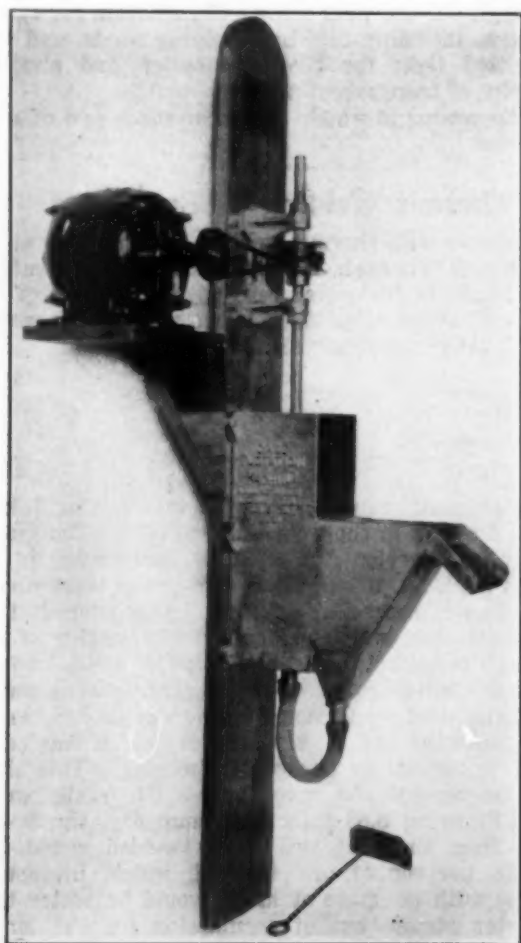


FIG. 1—FLOTATION TESTING MACHINE

timely and welcome addition to the laboratory of the metallurgical investigator. A recent modification of the type shown in the illustration has only a single-speed pulley on the driving shaft and a variable-speed motor. This combination is thought to offer some advantages.

Sampling Lead Bullion

The methods of sampling lead bullion at Salida, Col., and El Paso, Tex., are given in the following extracts from papers appearing in Bulletins of the A. I. M. E.

According to Mr. F. D. Weeks, an early method of sampling at Salida was to take a dip from the lead well of the furnace, but this was not satisfactory. The present management built drossing kettles in which the bullion is stirred by compressed air, skimmed into a Howard press, stirred by hand and siphoned out into bars for shipment, after sampling by dipping "gum-drops," approximately $\frac{1}{2}$ assay ton in weight. For some time the results were unsatisfactory. Although the copper had been reduced to 0.06 per cent, the bullion had a dirty appearance, and "gum-drops" of the same dip would not check with each other, especially as gold content. Consultation with men versed in the subject did not help. A remedy was finally found when the temperatures were determined by a pyrometer and the difficulties overcome by the following procedure: Take off heavy dross at 900 deg. Fahr.; blow until temperature reaches 680 deg. Fahr., skimming from time to time; raise the heat to 720 deg. Fahr. for sampling, and mold at 800 deg. Fahr. Since this method was adopted, little trouble has been encountered.

A plan to eliminate the personal equation in bullion assaying has been carried out for some years, with good results. The "gum-drops," weighing about $\frac{1}{2}$ assay ton each, are weighed without trimming, cupelled, and parted; the weights of the "gum-drops" in decimals of an assay ton, of the gold and silver together and of the gold alone, in milligrams, are reported by the assayer. The calculations are then made in the office on a machine. In this way the assayer is not tempted to make his results check, because until they are figured he does not know whether they will check or not. The excellent results obtained are sufficient reason for following the plan.

The method used at El Paso is described by H. F. Easter, as follows: Whether drossed at the blast furnace or at the remelting furnace, the bullion is sampled in the same manner, as follows: The bullion is tapped into a cast-iron cooling pot. The dross is skimmed off carefully and thoroughly. Before the final skimming the contents of the kettle are well stirred with the skimmer and the dross thereby raised is taken off completely. After skimming, the contents of the cooler are stirred again and, while the lead is still in motion, "gum-drop" samples are dipped. A few more gum-drops are taken than it is expected to make bars of lead, to insure having a gum-drop for each bar of lead. The bullion is ladled into molds, and after the pot is empty all extra gum-drops over the number of bars produced are thrown out. These gum-drops are held and delivered to the assay office whenever the bullion corresponding to them is removed.

A shipping lot consists of 780 bars (about 42 tons), and, when complete, the 780 gum-drops corresponding are placed in a graphite crucible and melted at as low a temperature as will give a homogeneous mass from which to dip the final gum-drop samples, upon which the actual assay of the lot is made. Great care is taken not to get the sample hot enough to oxidize the lead during the process.

The gum-drops, which weigh about 40 g. each, are assayed without clipping and the gold and silver calculated to an assay-ton basis.

A New Die Casting Metal

A new alloy which has recently been placed on the market under the name of "Ampco bronze" is unlike any other bronze in that it contains neither zinc, tin, lead or phosphorus. This new bronze was developed to supply the demand for a strong die-casting metal to be used in places where white metal die-castings were too weak and porous. Great difficulties have attended the attempts to form suitable die-casting copper-base alloys, as usually they melted at too high a temperature and attacked the dies when containing zinc and tin.

Ampco bronze is the result of ten years of research work, and is a 90 per cent copper-base alloy. It can be die-cast in permanent steel molds to finished dimensions and has great strength at high temperatures. This enables this metal to overcome the strains set up in the castings in the die molds while the metal is cooling and shrinking in the die.

This bronze is produced in several grades, differing slightly in composition, but considerably in physical properties. They have a range of tensile strengths of from 50,000 lb. per square inch to 100,000 lb. per square inch; elastic limit of 25,000 lb. per square inch to 45,000 lb. per square inch; elongation of 50 to 6 per cent; hardness of 70 to 280.

The metal is non-corrodible by acids, malleable, ductile, and can be forged, hammered, rolled or drawn. The tests given above are for sand castings. The forged, rolled and drawn metal has a tensile strength up to 150,000 lb. per square inch.

It is manufactured by the American Metal Products Company, Milwaukee, Wis.

Personal

Mr. W. R. Cox has been appointed professor of mining and metallurgy in the Oregon School of Mines, vice G. Montague Butler, resigned.

Mr. A. C. Dart has returned to Denver after an absence of several years and has again entered the professional field as a mining and metallurgical engineer.

Mr. John Gross has returned to Denver from Central America and opened his office as metallurgical engineer at 523 McPhee Building.

Mr. P. Kranefeld has removed from Montreal, Canada, where he was representing the Krupp concern, to Denver, Col., and will continue to specialize in crushing and grinding apparatus.

Mr. Phillip J. Krell will become manager of the new plant of the International Oxygen Company in the course of erection at Verona, Pa.

Mr. J. P. Mooney, formerly with the B. F. Goodrich Co., Akron, Ohio, has been appointed sales manager of the commercial department of the Hyatt Roller Bearing Co., Newark, N. J.

Mr. James J. Ormsbee, recently with the Tacoma and Hayden Smelters of the A. S. & R. Co., is spending a short time in New York.

Mr. W. Acheson Smith, general manager of the International Acheson Graphite Co., was elected on Nov. 2 one of the commissioners of the new commission government of Niagara Falls.

Dr. Walter O. Snelling of Pittsburgh, Pa., will erect an extensive laboratory in Long Island City, N. Y.

Dr. Charles P. Steinmetz was elected president of

the common council of Schenectady, N. Y., on the socialistic ticket, on Nov. 2.

Mr. W. R. Thurston has been appointed manager of the Mine & Smelter Supply Co.'s Salt Lake City branch.

Mr. Howard H. Utley is local manager for the River Smelting & Refining Co., a new concern which is remodeling the old pyritic smelting plant at Florence, Col., to suit the requirements of proposed operations. L. V. Emanuel is superintendent and George B. Clark chemist.

Mr. H. P. Zeller has resigned his position as assistant superintendent of the coke plant of the Republic Iron Steel Company and has become superintendent of the new by-product plant of the Toledo Products Company.

Obituary

Abraham Van Winkle, president of the Hanson & Van Winkle Company, Newark, N. J., died on Sept. 30. He had been active in the electroplating trade for over forty years, and his untiring energy had made him a commanding figure. Born in Pompton, N. J., he was first interested in the drug business in Newark, N. J., but in 1870 went into the plating field. He was interested with Dr. Edward Weston in the first low-voltage dynamo made in this country. This dynamo was exhibited in Philadelphia in 1876. Mr. Van Winkle was also connected at one time with the Brush Electric Company, and his successes were instrumental in the forming of the U. S. Electric Lighting Company. He manifested at all times a great interest in all the affairs of the electroplating trade as well as in electrochemistry in general. His loss is keenly felt by all who knew him.

Digest of Electrochemical U. S. Patents

PRIOR TO 1903

Arranged according to subject-matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

Aqueous Bath Apparatus

(Continued)

225,571, March 16, 1880, Patrick Concannon of Wallingford, Conn.

Relates to a rack for holding spoons and the like which are subjected to unequal wear, the spoons being held at the points of least wear. The holding clamp is mounted on a small shaft provided with a pinion which engages a movable toothed rack to oscillate the pinion, thereby dipping first the bottom of the bowl of the spoon in the electrolyte, then the underside of the handle, these portions receiving the greater amount of wearing use. Several spoon-holders are arranged in one frame to operate simultaneously. The device may be used for replating worn spoons.

234,775, Nov. 23, 1880, Robert B. Herskell of Wallingford, Conn., assignor to Simpson, Hall, Miller & Co. of same place.

Relates to a holder to support spoons and the like to be electroplated, and which are subjected to unequal wear. On opposite sides of the plating tank are arranged two uprights which support a cross-bar between them, the height of the cross-bar being adjustable. Upon this bar are fastened brackets to hold the spoons, the latter may therefore be immersed in the plating solution to any desired depth. Upon the remaining sides of the tank are arranged shelves supporting contact plates, the spoons resting on the contact plates and

the edge of the tank and extending into the solution, the outside of the bowl of the spoon dipping slightly in the solution so as to receive a deposit over a limited area.

245,206, Aug. 2, 1881, Hans J. Müller, of New York, N. Y., assignor to himself and Alexander Levett of same place.

Relates to protecting plating baths from reversing the polarity of the field magnets of a plating machine, and refers to his prior patent, No. 241,053, dated May 3, 1881, for a dynamo, which should be consulted with this for an understanding of the invention, which consists broadly in providing an extra circuit, using a carbon cathode, through which enough current is flowing to keep the fields excited.

248,436, Oct. 18, 1881, Thomas A. Edison, of Menlo Park, N. J., assignor to the Edison Electric Light Company, of New York, N. Y.

Relates to cells for depositing copper or the like upon the joints between the carbon filament and leading-in wires of an incandescent lamp. The cell consists of a tubular shaped body having a flared opening at the bottom, to receive a rubber stopper. The stopper is perforated to receive the glass tube of the lamp-stem supporting the leading-in wires, the stem being adjustable in the perforation to bring the joint just below the surface of the electrolyte. Instead of a cell consisting of a single tube, it may be a larger vessel having a plurality of openings in its bottom to receive stoppers. In the plating operation, the leading-in wires are connected as cathode, a suitable anode or anodes being suspended in the electrolyte. The electrolyte is drawn off when the plating operation is completed.

258,214, May 23, 1882, Alexander Brinckmann, of Astoria, assignor to himself and George H. Wooster of New York, N. Y.

Relates to supporting frames for small articles like "plate-pins" used in piano-fortes, and the like, such articles being too small to be held in any of the usual clamps. The supporting frames consist of one or more sheets of metal, preferably a plurality, the sheets being secured in parallel planes by metal supports at the corners, the bottom sheet having a plurality of small irregularly shaped perforations too small for the pin to fall through and making contact at points; the upper sheets being perforated with registering holes larger than the pin, to allow circulation of the electrolyte around the pin. In use pins are inserted in the aligned holes, and the frame immersed by wires or rods in the electrolyte, and connected as cathode.

297,669, April 29, 1884, Charles F. Brush of Cleveland, Ohio.

Relates to apparatus for electroplating carbons for arc lighting, etc., and refers to his prior patent No. 196,425, which relates to the advantages of electroplated carbons. The present apparatus consists of a plurality of individual depositing cells connected in series, each to receive one carbon connected as cathode, and having a tubular anode. A tube to supply compressed air is also inserted in each cell, the compressed air stirring the electrolyte. The inventor finds it necessary to connect the cells in series, using a high potential current to supply the current necessary for plating, on account of the irregularities in individual carbons, these irregularities causing uneven and irregular deposits when the carbons are connected in parallel as usual.

310,146, Dec. 30, 1884, Arthur Murphy of Taunton, Mass., assignor to himself and Charles Minchew.

Relates to baskets and the like made of glazed earthenware, etc., and having perforated sides and bottom. Attached to the bottom are a plurality of metallic buttons electrically connected to each other and to a metallic

suspending hook attached to the bail or handle of the basket. The apparatus is for electroplating tacks and other small articles.

331,609, Dec. 1, 1885, Herman R. Boissier of New York, N. Y.

Relates to methods of connecting the electroplating tanks to a dynamo. With a series dynamo, a small quantity of work in the plating bath does not permit enough current to pass to supply the field magnets, consequently the plating operation is exceedingly slow. The inventor provides an extra tank connected across the brushes of the dynamo, the extra tank containing an anode and cathode, and may be used for quick work; when not thus used it serves to produce electrolyte by the solution of the anode. The field magnet coils are provided with several shunts which may be used to vary the current flowing through the fields when the principal plating tank is filled with work.

369,457, Sept. 6, 1887, Morris H. Levett of New York, N. Y., assignor of one-third to Louis Levett.

Relates to electroplating tanks, and is for a folding removable inner cell made of fabric supported by wire mesh sides and bottom. The fabric cell is so made that its sides may be folded inward and entirely cover the bottom. In use, a plating tank accumulates a lot of scum and sediment, which renders the solution turbid. This sediment settles on the fabric bottom; the fabric sides are then folded down to inclose it, the fabric cell then lifted from the tank and cleaned. The inner cell thus serves to maintain a clean electrolyte.

Book Reviews

The Analysis of Non-Ferrous Alloys. By F. Ibbotson and L. Aitchison. Octavo, 230 pages, 19 illus. Price, \$2.25. London and New York: Longmans, Green & Company.

This is intended for laboratory instruction as well as for use by the works' chemist. It is admirably adapted for both purposes, being keenly analytical in its method of presentation, explaining clearly the *why* and *wherefore* and the scientific basis for the methods described. It is up to date, as seen by it describing the analysis of the new high-speed tool alloy "stellite." It is, however, limited to the determination of lead, copper, bismuth, antimony, tin, arsenic, aluminium, chromium, nickel, cobalt, manganese (iron) and zinc. The other non-ferrous metals and non-metals occurring in alloys (such as cerium, silicon, phosphorus, titanium, molybdenum, etc.) are not considered. Within the limits treated it is a most admirable book.

Principles of Physical Chemistry.—By Edward W. Washburn. Octavo, 445 pages, 61 illustrations. Price \$3.50. New York: McGraw-Hill Book Company, Inc.

The sub-title states that the treatment is from the standpoint of modern atomistics and thermodynamics, and that it is intended for students intending to enter physics or chemistry as a profession. It is not a laboratory manual, but a thorough discussion based on first principles, most admirably planned and very skillfully executed. The mastery of the author over every phase and topic is in evidence throughout the book, giving to the reader a sense of confidence in the information and lending a charm to the reading which is as refreshing as it is unusual. The book pre-supposes a knowledge of general chemistry and physics, and in mathematics the elements of differential and integral calculus. Anyone with this equipment will find this book probably the best help he can get in print to a thorough acquaintance with and a working knowledge of the principles of physical chemistry.